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PREFACE.

The papers and discussions presented at meetings of the Institute during the year 1915 will be published in three volumes. The present volume contains the papers and discussions presented at the New York Meeting, held in February, 1915.

Vol. LII will be devoted to the papers and discussions presented at the San Francisco Meeting, September, 1915. Since it will not be practicable to include in this volume all of the papers presented at that meeting, some of them, probably those on iron and steel and related subjects, will be held over and published in Vol. LIII, with papers on similar subjects read at the February, 1916, meeting.

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JAMES B. RISQUE,
MILNOR ROBERTS,
OSCAR ROHN,
HENRY L. SMYTH,
BENJAMIN W. VALLAT,
RUEL C. WARRINER,
SAMUEL D. WARRINER,
GEORGE WEIR,
DWIGHT E. WOODBRIDGE.

Milling Methods

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WILLIAM A. CALDECOTT,
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H. A. GUESS,
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HENRY S MUNROE,
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LEWIS G. ROWAND,
E. A. C. SMITH,
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WALTER H. WILEY.

The Use of Electricity in Mines

WILLIAM KELLY, *Chairman*.

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OZNI P. HOOD,
JOHN LANGTON,
THOMAS H. LEGGETT,
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STEPHEN H. PITKIN,
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SOLOMON LeFEVRE,
WILLIAM W. MEIN,
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JAMES B. RISQUE,
FRANCIS P. SINN,
WILLIAM D. THORNTON,
WILLIAM R. WALKER,
C. W. WHITLEY,
HENRY A. J. WILKENS,
DWIGHT E. WOODBRIDGE.

Proceedings of the One Hundred and Tenth Meeting, New York City, February, 1915

Local Committees

General Committee of Arrangements.—L. W. Francis, Chairman; Bradley Stoughton, Vice-Chairman; Louis D. Huntoon, John H. Janeway, Willard S. Morse, P. A. Mosman, George C. Stone, E. Maltby Shipp, W. L. Saunders, Charles F. Rand.

Ladies' Committee.—Mrs. James F. Kemp, Chairman; Mrs. H. A. Prosser, Vice-Chairman; Mrs. R. M. Atwater, Mrs. George D. Barron, Mrs. Charles A. Bohn, Mrs. J. H. Devereux, Mrs. Walter B. Devereux, Jr., Mrs. Arthur S. Dwight, Mrs. Karl Eilers, Mrs. Lewis W. Francis, Mrs. John Hays Hammond, Mrs. H. W. Hardinge, Mrs. Levi Holbrook, Mrs. Louis D. Huntoon, Mrs. Axel O. Ihlseng, Mrs. W. R. Ingalls, Mrs. John H. Janeway, Mrs. Sidney J. Jennings, Mrs. Albert R. Ledoux, Mrs. W. W. Mein, Mrs. Willard S. Morse, Mrs. Henry S. Munroe, Mrs. Charles F. Rand, Mrs. Thomas T. Read, Mrs. Burr A. Robinson, Mrs. Thomas Robins, Mrs. E. Maltby Shipp, Mrs. Bradley Stoughton, Mrs. Benjamin B. Thayer, Mrs. Leonard Waldo.

Annual Dinner Committee.—William L. Saunders, Chairman; W. de L. Benedict, Arthur S. Dwight, Henry M. Howe, John C. Montgomery, E. Maltby Shipp, Edward B. Sturgis.

Luncheons Committee.—E. Maltby Shipp, Chairman; W. de L. Benedict, Charles A. Bohn, Arthur S. Dwight, P. A. Mosman, Lucius W. Mayer, Carr B. Neel.

Program Committee.—George C. Stone, Chairman; Robert M. Catlin, John V. N. Dorr, Louis C. Graton, J. E. Johnson, Jr., James F. Kemp, Joseph W. Richards, Bradley Stoughton.

Reception Committee, Monday, February 15, 1915.—A. R. Ledoux, Chairman; R. M. Atwater, Jr., Charles P. Berkey, F. Lynwood Garrison, E. H. Gary, N. V. Hansell, John H. Janeway, Hennen Jennings, George F. Kunz, J. S. Lane, Richard Moldenke, Rossiter W. Raymond, Burr A. Robinson, John D. Ryan, Theodore Sternfeld, Edward L. Young.

Reception Committee, Tuesday, February 16, 1915.—Sidney J. Jennings, Chairman; Sidney H. Ball, C. F. Chandler, Christopher R. Corning, John H. Hall, Joseph A. Holmes, W. R. Ingalls, J. E. Johnson, Jr., Charles Kirchhoff, Thomas H. Leggett, Robert H. Richards, Denis M. Riordan, George C. Stone, Bradley Stoughton, William D. Thornton, Horace V. Winchell.

Reception Committee, Wednesday, February 17, 1915.—Joseph Struthers, Chairman; George D. Barron, C. C. Burger, Warren Delano, John V. N. Dorr, James Gayley, Levi Holbrook, James F. Kemp, Walde-

mar Lindgren, A. F. Lucas, H. A. Prosser, Joseph W. Richards, Kirby Thomas.

Technical Sessions

Monday Morning, February 15, 1915.—The opening session, like all the technical sessions, was held at the headquarters of the Institute in the Engineering Societies' Building, New York City. The opening session was under the auspices of the Committee on Safety and Sanitation. Mr. Arthur Williams, Chairman of that Committee, presided.

In connection with this session, an exhibition on safety and sanitation was maintained by the Committee during all four days of the meeting, in the rooms of the American Museum of Safety on the fifth floor of the Engineering Societies Building.

The following papers were presented by their authors or authors' representatives:

Safeguarding the Use of Mining Machinery, Frank H. Kneeland. Discussed by Arthur Williams, William H. Tolman, Benjamin F. Tillson.

Safety Methods and Organization of United States Coal & Coke Co., Howard N. Eavenson. Discussed by William H. Grady, Thomas W. Dawson (written discussion), Charles W. Goodale, J. Parke Channing, B. F. Tillson, George S. Rice, Arthur Hovey Storrs, Harry H. Stoek, Lawson Blenkinsopp (written discussion), Arthur Williams, Howard N. Eavenson.

Housing and Sanitation at Mineville, S. Le Fevre.

Enlarging the Worth of the Worker and the Perspective of the Employer, J. Parke Channing. Discussed by Fred H. Rindge, Jr., S. A. Taylor, Harry H. Stoek, Arthur Williams.

Cost Factors in Coal Production, William H. Grady. This paper was discussed at a special session held on Wednesday afternoon, February 17, at which Samuel A. Taylor presided. Discussed by S. A. Taylor, George S. Rice, H. M. Crankshaw, Edwin Ludlow, Charles Enzian, Arthur Hovey Storrs, W. G. Whildin, Edward W. Parker, William H. Grady.

Monday Afternoon, February 15, 1915.—This session was under the auspices of the Committee on Non-Metallic Minerals, and was presided over by Dr. Heinrich Ries, and by Professor Charles P. Berkey while Dr. Ries occupied the floor.

The following papers were presented by their authors or authors' representatives:

Barite of the Appalachian States, Thomas L. Watson and J. Sharshall Grasty.

German and Other Sources of Potash Supply, Charles H. MacDowell. Discussed by George S. Rice, V. C. Grubnau, Charles H. MacDowell.

The Plasticity of Clay and Its Relation to Mode of Origin, N. B. Davis.

White-Burning Clays of the Southern Appalachian States, Joel H.

Watkins. Discussed by Richard H. Vail, John J. Blow, Sidney J. Jennings, H. Ries, Joel H. Watkins.

The following papers were presented by title:

Investigation of Sources of Potash in Texas, William B. Phillips.

The Origin of the Louisiana and East Texas Salines, Edward G. Norton.

Monday Afternoon, February 15, 1915.—This session was under the auspices of the Iron and Steel Committee, Professor Albert Sauveur presiding.

The following papers were presented by their authors or authors' representatives.

High Blast Heats in Mesaba Practice, Walther Mathesius. Discussed by Joseph W. Richards, W. A. Forbes, J. E. Johnson, Jr., H. P. Howland (written discussion).

Modern Gas-Power Blower Stations, Arthur West. Discussed by Joseph W. Richards, J. E. Johnson, Jr., Arthur West.

Effect of Finishing Temperatures of Rails on Their Physical Properties and Microstructure, W. R. Shimer. Discussed by William R. Webster, Henry M. Howe, A. A. Stevenson, G. K. Burgess.

Sound Steel Ingots and Rails, George K. Burgess and Sir Robert A. Hadfield. Discussed by Albert Sauveur, Henry M. Howe, Henry D. Hibbard, Joseph W. Richards, George K. Burgess.

Are the Deformation Lines in Manganese Steel Twins or Slip Bands? Henry M. Howe and Arthur G. Levy. Discussed by Albert Sauveur.

The Mayari Iron-Ore Deposits, Cuba, J. F. Kemp.

The following papers were read by title:

Structure and Hysteresis Loss in Medium-Carbon Steel, F. C. Langenberg and R. G. Webber.

The Electric Furnace in the Foundry, William G. Kranz.

The Duplex Process of Steel Manufacture at the Maryland Steel Works, F. F. Lines.

Tuesday Morning, February 16, 1915, at 11 *a.m.*—Professor Robert H. Richards presiding.

The following papers were presented by their authors or authors' representatives:

Experiments on the Flow of Sand and Water Through Spigots, Robert H. Richards and Boyd Dudley, Jr. Discussed by R. H. Richards, Albert R. Ledoux.

The Boulder Batholith of Montana, Paul Billingsley. Discussed by James F. Kemp, Darsie C. Bard (written discussion), Waldemar Lindgren, Horace V. Winchell, L. C. Graton, Paul Billingsley.

Metallurgical Practice in the Porcupine District, Noel Cunningham. Discussed by Robert H. Richards.

The following papers were presented by title:

Development of the Butchart Riffle System at Morenci, David Cole. Discussed by R. H. Richards, Edward P. Mathewson.

The Main Mineral Zone of the Santa Eulalia District, Chihuahua, Basil Prescott.

The Cloncurry Copper District, Queensland, W. H. Corbould.

The Mining and Reduction of Quicksilver Ore at the Oceanic Mine, Cambria, Cal., C. A. Heberlein. Discussed by Hugh D. Pallister (written discussion).

An Improved Form of Cam for Stamp Mills, Arthur B. Foote.

Method for the Determination of Gold and Silver in Cyanide Solutions Luther W. Bahney.

Tuesday Afternoon, February 16, 1915.—This session was under the auspices of the Committee on Petroleum and Gas and the Committee on Coal and Coke, and was presided over by Captain Anthony F. Lucas.

The following papers were presented by their authors or authors' representatives:

Recent Developments in Coal Briquetting, Charles T. Malcolmson. Discussed by E. W. Parker, Alfred C. Lane, Charles Dorrance, Jr., Edwin Ludlow.

Depreciation as Applied to Oil Properties, Philip W. Henry. Discussed by C. E. Grunsky, Jr. (written discussion).

The Use of Mud-Ladened Water in Drilling Wells, I. N. Knapp. Discussed by A. Beebe Thompson (written discussion), Henry Louis (written discussion), A. C. Lane.

The Rôle and Fate of the Connate Water in Oil and Gas Sands, Roswell H. Johnson. Discussed by A. C. Lane, D. B. Reger, I. N. Knapp, David T. Day, Roswell H. Johnson.

The Petroleum Fields of Alaska, Alfred H. Brooks.

A Modern Rotary Drill, Howard R. Hughes. Discussed by I. N. Knapp.

The Dehydrating Oil Plant of Nevada Petroleum Co., California, S. J. Hardison.

Contributions from the Oil and Gas Production Laboratories of the University of Pittsburgh. Discussed by David T. Day, Roswell H. Johnson.

The following papers were presented by title:

The Limits of Mining under Heavy Wash, Douglas Bunting.

Comparative Costs of Rotary and Standard Drilling, Mark L. Requa.

Improved Methods of Deep-Drilling in the Coalinga Oil Field, California, M. E. Lombardi. Discussed by I. N. Knapp.

The Estimation of Oil Reserves, Chester W. Washburne. Discussed by Roswell H. Johnson.

Oil and Gas Possibilities of Kentucky, F. Julius Fohs.

Gasoline from "Synthetic" Crude Oil, Walter O. Snelling. Discussed by A. F. Lucas, William N. Best, F. G. Cottrell, M. R. Wolfard, David T. Day, Leonard Waldo, Roswell H. Johnson, Walter O. Snelling.

Wednesday Morning, February 17, 1915.—This session was under the auspices of the Committee on Precious and Base Metals. Charles W. Goodale presided at first, and was followed by Edward P. Mathewson.

The following papers were presented by their authors or authors' representatives:

Gold-Bearing Gravels of Beauce County, Quebec, J. B. Tyrrell. Discussed by Edward P. Mathewson.

A Study of the Chloridizing Roast and Its Application to the Separation of Copper from Nickel, Boyd Dudley, Jr. Discussed by H. O. Hofman.

Effect of Zn_3Ag_2 upon the Desilverization of Lead, F. C. Newton. Discussed by H. O. Hofman, J. W. Richards, W. McA. Johnson, Edward P. Mathewson, F. C. Newton.

Copper Smelting in Japan, Manual Eissler. Discussed by J. W. Richards.

Coal-Dust Fired Reverberatory Furnaces of Canadian Copper Co., David H. Browne.

Coal-Dust Fired Reverberatories at Washoe Reduction Works, Louis V. Bender.

Reverberatory Smelting Practice of Nevada Consolidated Copper Co., R. E. H. Pomeroy. These last three papers were discussed together by Edward P. Mathewson, Charles W. Goodale, David H. Browne, N. M. Langdon, James B. Herreshoff, W. McA. Johnson, J. W. Richards, Henry D. Hibbard, Frank Klepetko, Albert F. Schneider, F. C. Newton, H. S. Munroe (written discussion), W. D. Leonard (written discussion).

Wednesday Afternoon, February 17, 1915.—This session was under the auspices of the Committee on the Use of Electricity in Mines and was presided over by William Kelly.

The following papers were presented by their authors or authors' representatives:

The Hydro-Electric Development of the Peninsular Power Co., Charles V. Seastone.

Underground Haulage by Storage-Battery Locomotives in the Bunker Hill & Sullivan Mine, J. W. Gwinn. Discussed by Girard B. Rosenblatt (written discussion), John Langton, H. M. Crankshaw, H. H. Clark, Charles W. Goodale, George R. Wood.

The Testing and Application of Hammer Drills, Benjamin F. Tillson. Discussed by T. E. Sturtevant, H. M. Crankshaw, Benjamin F. Tillson.

Some Defects of the United States Mining Law, Courtenay De Kalb. Discussed by Horace V. Winchell.

The following papers were presented by title:

Mining Methods of the Arizona Copper Co., P. B. Scotland.
Mining Methods of Park City, Utah, James Humes.

Entertainment of Ladies

The Ladies' Committee assembled in the Reception Room on the first floor each day of the meeting from 12 to 1 o'clock to receive visitors and escort them to luncheon. The presence of many ladies at the luncheons was an especially enjoyable feature of the meeting. The entertainment of the ladies comprised a concert on Monday afternoon followed by a Thé Dansant at the Hotel Biltmore, at which the famous Mr. and Mrs. Maurice gave an exhibition of dancing. On Tuesday afternoon the ladies were taken by automobiles to the Metropolitan Museum of Art, and many ladies were in attendance at the Annual Dinner that evening. On Wednesday afternoon Senator and Mrs. W. A. Clark opened their home to all members and guests of the Institute, and an opportunity was afforded to the visiting ladies to view the art treasures and to listen to music on the pipe organ.

Social Features of the Meeting and Visits

Annual Dinner.—The Annual Dinner was held on Tuesday evening, February 16, at the Hotel Astor. The attendance included 296 members and guests. The dinner was preceded by a reception at 6:30, and after-dinner addresses were made by the following speakers: Benjamin B. Thayer, Gano Dunn, Job E. Hedges, Rossiter W. Raymond, John C. Montgomery, William L. Saunders.

New York Club Privileges.—The following New York Clubs very courteously extended their facilities to visiting members during the meeting and honored cards of introduction from the Institute:

Chemists' Club	Machinery Club	Technology Club
Columbia University Club	Princeton Club	Williams Club
Harvard Club	Rocky Mountain Club	Yale Club

Luncheons.—Luncheons were served on each of the three days in buffet fashion in the room adjoining that in which the technical sessions were held.

Meetings of College Alumni.—Correspondence was had between the Local Committee of the Institute and some 22 colleges, each of which appointed a committee to communicate with the graduates of the college who are members of the Institute, to urge their attendance at the meeting and to facilitate gatherings of the graduates. This resulted in a number of informal gatherings, including luncheons, dinners and so forth, by the graduates of the different colleges as well as arrangements by several groups to sit together at the Annual Dinner.

Excursion to New Subway.—About 50 members of the Institute were afforded an opportunity to inspect some of the construction work in progress on the New Subway on Thursday, Feb. 18, 1915. The party was taken in automobiles to many different points, and an excellent opportunity given to see the very interesting work in progress. The thanks of the Institute are due particularly to J. F. Sanborn, who arranged for the excursion, and Robert Ridgway, Engineer in charge of Subway Construction of the First District for the Public Service Commission, who personally conducted the party on its trip of inspection. At the various points at which stops were made, the division engineers and their assistants pointed out to the party features of the work in their particular sections.

Co-operation with American Institute of Electrical Engineers.—Since the American Institute of Electrical Engineers held their Annual Meeting on February 17 to 19, 1915, an opportunity for co-operation was afforded: the members of the A. I. E. E. were invited to attend the sessions of the A. I. M. E. on Wednesday, February 17, dealing with Electricity in Mines and Mining, while the members of the A. I. M. E. were invited to attend the session of the A. I. E. E. on Friday, February 19, at which papers on the Cottrell Process for Electrical Precipitation were presented and discussed.

Delegates from Other Societies.—The following Societies were represented by delegates at the meeting of the Institute:

American Chemical Society, Joseph W. Richards.

American Electrochemical Society, E. F. Roeber and Joseph W. Richards.

American Institute of Electrical Engineers, F. L. Hutchinson.

The American Museum of Safety, Arthur Williams and Frederick Remsen Hutton.

The American Society of Heating and Ventilating Engineers, Arthur Ritter, W. W. Macon and W. S. Timmis.

Canadian Mining Institute, H. Mortimer-Lamb.

American Society for Testing Materials, Edgar Marburg.

The Franklin Institute, R. A. F. Penrose, Jr. and Edward V. d'Invilliers.

The Geological Society of America, Henry Fairfield Osborn, James F. Kemp and George F. Kunz.

New York Electrical Society, George H. Guy.

The Society for Electrical Development, Theodore Dwight.

Registration and Membership Facilities at Headquarters.—Registration facilities were provided for all three days of the meeting and the total number of members and guests registered was 357. Special registration facilities were also provided to enable graduates of the different colleges to arrange for gatherings.

The usual facilities of the Institute headquarters were also available,

including receipt and despatch of mail, telegrams, telephone messages, express and messenger service, etc., free dressing rooms for men, opportunities for checking articles, writing-room facilities and an information bureau. A special opportunity was also given for the private dictation of discussions for the benefit of those who wished to prepare themselves for presenting discussion, or who wished to discuss in writing instead of verbally.

Meetings of Committees.—The following Committees held meetings during the three days' session of the Institute, the list being given in the chronological order in which the meetings were held:

Committee on Safety and Sanitation.
Committee on Increase of Membership.
Committee on Non-Metallic Minerals.
Committee on Iron and Steel.
Committee on Mining Geology.
Committee on Milling Methods.
Committee on Coal and Coke.
Committee on Petroleum and Gas.
Committee on Junior Members and Affiliated Student Societies.
Committee on Precious and Base Metals.
Committee on Use of Electricity in Mines.
Committee on Mining Methods.
Committee on Mining Law.

PAPERS

The Mayari Iron-Ore Deposits, Cuba

BY J. F. KEMP, NEW YORK, N. Y.

(New York Meeting, February, 1915)

Introduction

THE *Bulletin* of the Institute for March, 1911, is chiefly devoted to papers upon the iron ores of northeastern Cuba. At that time information about the new developments in the peculiar brown hematites of the region was becoming widely circulated and interest was especially keen. For some years exploration had been conducted with pits and trenches and so great an area had been shown to contain ore in the Mayari, Moa, and Cubitas or San Felipe districts, that the reserves conveniently situated for the consumption of American furnaces were estimated as quite two billions of tons. In the subjoined footnote will be found a list of the principal papers of special interest which have already been published.[1] The further remark may, however, be made that the

[1] C. W. Hayes, T. W. Vaughan, and A. C. Spencer: Report on a Geological Reconnaissance of Cuba, under the Direction of Gen. Leonard Wood, Military Governor. The work in the Mayari, Moa, and Cubitas areas seems to have been chiefly done by Mr. Spencer, pp. 28, 83, 84 (1901).

Anon: The Mayari Iron-Ore District of Cuba. *The Iron Age*, vol. lxxx, pp. 421 to 426 (Aug. 15, 1907).

Anon: Iron Mining in Cuba. *The Iron Age*, vol. lxxxi, pp. 1149 to 1157 (Apr. 9, 1908).

A. C. Spencer: Three Deposits of Iron-Ore in Cuba. *Bulletin No.* 340, *U. S. Geological Survey*, pp. 318 to 329 (1908).

C. M. Weld: The Residual Brown Iron-Ores of Cuba. *Trans.*, xl, 299 to 312 (1909).

J. S. Cox, Jr.: Iron-Ores of the Moa District, Oriente Province, Island of Cuba. *Trans.*, xlii, 79 to 90 (1911).

W. L. Cumings and B. L. Miller: Characteristics and Origin of the Brown Iron-Ores of Camaguey and Moa, Cuba. *Trans.*, xlii, 116 to 137 (1911).

C. W. Hayes: The Mayari and Moa Iron-Ore Deposits of Cuba. *Trans.*, xlii, 109 to 115 (1911).

C. K. Leith and W. J. Mead: Origin of the Iron-Ores of Central and Northeastern Cuba. *Trans.*, xlii, 90 to 102 (1911).

J. E. Little: The Mayari Iron-Mines, Oriente Province, Island of Cuba, as Developed by the Spanish-American Iron Co. *Trans.*, xlii, 152 to 169 (1911).

A. C. Spencer: Occurrence, Origin and Character of the Surficial Iron-Ores of Camaguey and Oriente Provinces, Cuba. *Trans.*, xlii, 103 to 109 (1911).

D. E. Woodbridge: Exploration of Cuban Iron-Ore Deposits. *Trans.*, xlii, 138 to 152 (1911).

nature of the ore has been the subject of an interesting litigation, because under the peculiar property rights inherited by the Cuban government from the previous Spanish authorities, a distinction is made between pigments and iron ores. The necessity therefore arose for deciding that these deposits were iron ore rather than raw materials for paint. The decisions of the Cuban courts have now interpreted them to be actually iron ores. The testimony of a number of geologists, specially qualified to speak authoritatively on this question, was sought for the judicial hearings and thus some of the earlier special studies, as printed in the *Bulletin* and *Transactions*, were made.

In summary, the published descriptions already impart to all readers the location of one great district near the town of Mayari, southeast of Nipe Bay, in northeastern Cuba; of a second, 40 miles and more farther east, called the Moa district; and of a third, about 160 miles to the northwest of Mayari, called the Cubitas in the papers of Cox and Spencer, but the San Felipe by Cumings and Miller. Still other areas of the red ferruginous soil are apparent in portions of western Cuba, but in most, if not in all, cases, they are so far from the sea, or so valuable for agriculture, as not to have attracted attention as a source of iron. Goodly percentages of iron are of course primarily necessary, coupled with absence of objectionable components, and so situated as to give transportation to American smelting centers at low freights. The chief interest at present is therefore attached to the Mayari district, as it is the only one as yet in active operation. In the Mayari district the ore is 15 miles from salt water, but when reached it forms a continuous area. In the Moa district, the ore-bearing ground even runs down to the seacoast in places, but as a whole it is more dissected by erosion, so that the productive area is cut up into separate blocks. The San Felipe (or Cubitas) is farther from a deep harbor than either of the others. In all the three districts the ore mantles the tops of relatively elevated areas. In the Mayari it covers a plateau ranging about 1,800 ft. above tide. In the Moa district the ore near the coast is on the tops of hills of moderate altitude but rises to the south. In the San Felipe it covers a plateau at 450 to 500 ft. above the sea.

History

Crusts and concretionary masses of brown iron ore were early noted in northeastern Cuba. J. S. Cox, Jr., writing in 1911 (see citation, p. 79), states that claims were located upon them in the Moa district more than 20 years earlier. A. C. Spencer remarked them in 1901 as appearing in red clay, but no one seems to have realized until three or four years later that the entire mass was high enough in iron to be an ore. Under J. S. Cox, Jr., explorations were begun in the Mayari district in 1904 upon the crusts or "plancha," and then analyses revealed the fhct that

not only the upper, dull-red portion of the so-called clay was valuable, but also the lower yellow parts as well. The construction of the Mayari plant of the Spanish-American Iron Co. was begun in 1907 and was completed in December, 1909. Shipments have been active ever since. The plant and mines were well described with maps and views by J. E. Little in 1911, as cited above. The chief changes since then have been the greater extent of the pits on the plateau at Woodfred, and the improvements in the village of Felton, the shipping port on Nipe Bay. The ore is principally treated at Sparrows Point, Md., and Steelton, Pa. It is shipped both in the crude state and as so-called "nodulized ore," or the product of kilns similar to modern cement kilns, in which the water, both absorbed and combined, is driven off and the fine ore is half fused or fretted into nodules. Mushiness in the stack is thereby prevented and the rather heavy percentage of water (25 per cent. absorbed and 14 per cent. combined) is driven off, to the diminution of freight charges. Three or four per cent. of water is again absorbed in the cooling vats into which the kilns discharge. Analyses of the raw ore are customarily made on samples dried at or above 100° C. They have averaged by the year: Fe, 48 to 49; Ni, 1; SiO_2, about 3; Cr, 1 to 2; Al_2O_3, 11 to 11.5; combined H_2O, 13 to 13.5. The nodulized ore runs: Fe, 55 to 56; Ni, 1 to 1.2; SiO_2, 4 to 4.4; Cr_2O_3, slightly more than the raw ore; Al_2O_3, 13 to 14; absorbed water, 3 to 3.5. In both, sulphur and phosphorus are negligible. In time, the great purity of these ores, combined with their percentages in nickel, and their convenient shipment from deep-water docks, should win a European as well as an American market.

Previous Work on the Geology

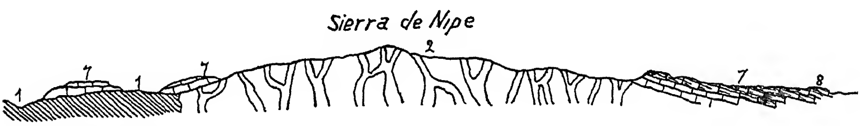
To the report of Messrs. Hayes, Vaughan, and Spencer, made in 1901 to Gen. Leonard Wood, at that time the Military Governor of Cuba, during the American occupation, we owe the only available systematic description of the island. As Messrs. Vaughan and Spencer were on the island but 12 weeks and Mr. Hayes but five, the report contains an extraordinary amount of detail when one recalls that the climate is tropical. The three geologists were chiefly busied with the mineral resources, but they give much information regarding general formations and made a careful summary of such previous literature as existed. Nevertheless, the report only whets our appetites for a full and accurate geological map and description of the island. Were it possible for the U. S. Geological Survey or some of our Academies of Sciences to make a co-operative survey with the Cuban government, as is now being done in Porto Rico in co-operation with the Porto Rican government, by the New York Academy of Sciences and affiliated New York City institutions, a great service would be rendered Cuba, as well as general science. Messrs.

Hayes and Spencer made a trip of eight days with a pack train from Santiago to Mayari and return. They crossed the tract of the Mayari mines and have drawn a geological section, which is here reproduced from their Plate XVIII, Section 6, in so far as it relates to the Sierra Nipe, on which the ore lies (Fig. 1).

The following quotations are taken from the report:

P. 28. "Along the line of the trail which crosses the Sierra Nipe leading from San Luis to Mayari, near the line of the section presented in Plate XVIII, Fig. 6, there is a belt of metamorphic and dense igneous rocks, between the Oligocene limestone and the serpentine which forms the axis of the Sierra. This series is very poorly exposed, but is probably older than the serpentine. Indications of a similar series were observed near the edge of the serpentine on the south side of the Sierra Cristal in the vicinity of Mayari Arriba.

"The main mass of the Sierra Nipe is composed of serpentine, which is shown by the microscope to have originated through the alteration of a rock originally composed largely of bronzite, fragments of which mineral are still frequently to be observed in the otherwise completely altered mass. Since this chemical alteration occurred, the serpentine has been intruded by dense diabase. These last-named rocks occur in



1, Metamorphic rocks; 2, Serpentine with diabase and gabbro intrusions; 7, Massive limestone (Oligocene?); 8, Coastal soborruco and Oligocene marls.

Fig. 1.—Reproduction of Northern Part of Geological Section, Plate xviii, Fig. 6, of Report by Hayes and Spencer.

such abundance that it is impossible to find any large area of the serpentine which is not cut at least by small dikes of the later rock, and frequently it is found in very large masses.

"On the northern side of the Sierra Nipe the Oligocene limestone reaches an elevation of about 900 ft., dipping northward toward the sea. In the higher exposures the dips are between 10° and 15°, but become less as the coast is approached."

On pp. 83 to 84 the following remarks appear with reference to the shot-like masses and crusts of limonite now called plancha:

"Occupying the general region between Nipe Bay and Moa Bay, and somewhat back from the northern coast, there is a region reaching a general elevation of from 1,500 to 2,000 ft., and occupied by serpentine and other igneous rocks. Upon the top of this sierra there are many large areas which are practically level, and these are always covered by a thick mantle of red clay, which contains a large proportion of iron ore in the form of spherical pellets. Locally this material entirely replaces the clay, and the separate particles are cemented together by ferruginous materials, making a spongy mass of brown iron ore. Similar occurrences of shot and massive ore were noted upon the tops of certain hills lying to the north of the city of Puerto Principe, and following the general trend of the Sierra Cubitas. The rock in this vicinity is also serpentine, and the ores have identical characteristics with those of the

region mentioned above. Analyses were made of these residual ores collected near Rio Seco along the trail between Mayari and San Luis."

The following two analyses are given with two others from localities of no interest in this connection:

	1 Per Cent.	2 Per Cent.
Moisture	....	0.56
Iron	52.00	54.69
Manganese	0.364	0.594
Phosphorus	0.0368	0.0189
Silica	2.62	2.51
Chromium	tr.	present
Titanium	0.25	....

1. Iron ore from Sierra Nipe near trail crossing the Rio Naranjo, about 10 miles from Mayari, Santiago Province (now Oriente Province).

2. Iron ore from Sierra Nipe near Rio Seco, Santiago Province (now Oriente Province).

"These residual ores are locally known as 'tierra de perdigones' or 'moco de herrero,' signifying shot-soil and blacksmith's waste, either of which terms is a very apt designation. Rodriguez Ferrer is authority for the statement that hydrated oxide of iron in the form of pellets in the soil occurs at various points in the Island. The following localities are mentioned: Province of Pinar del Rio, between Consolacion del Sur and Candelaria; Matanzas Province in the Sierra Morena, between Cardenas and Sagua la Grande; Loma Iman near the city of Puerto Principe, and Monte Libano north of Guantanamo, Santiago Province. The amount of these ores in various parts of the island is certainly very large and it seems not improbable that they may eventually find a market in the United States in cases where they are situated near a sufficient supply of running water for washing them free from the clay with which they are mixed."

That the mantle of surface materials was due to the alteration or weathering of serpentine was recognized by Hayes and Spencer, and some notes on the mother rock of the serpentine were made, based on microscopic study. They also noted the later dikes described as diabase, presumably observed in ascending and descending the plateau, since on the top everything was then concealed. So far as the ore is concerned it only remained for Mr. Cox and his engineers to discover that the "clay" was also iron ore.

All the other observers corroborate the derivation of the ore by the weathering of serpentine, and Mr. Spencer in his paper of 1908 goes more at length into this subject, and notes the parallels with the ores formerly dug on Staten Island, N. Y., and with the deposits known to exist at Clealum, Wash. C. M. Weld in 1909 takes up the process of tropical weathering much more fully and acutely recognizes the similarity of the ore to the laterites of India. Mr. Weld also gives four fairly complete analyses from Mayari, Moa, Taco, and Navas, which he had evidently recast into their possible minerals because he states, p. 302: "Within the clay-ore are found disseminated nodules and pellets of brown

ore ranging apparently through all the hydrated forms from limonite to turgite; hematite also is present and at times magnetite." Mr. Weld obviously recognized that the combined water was too small to satisfy the alumina for bauxite ($Al_2O_3.2H_2O$) and the iron for limonite ($2Fe_2O_3.3H_2O$). The same point is emphasized by C. K. Leith and W. J. Mead and will be more fully discussed later on. Mr. Weld gives the first published analysis of the serpentine, placing by its side a reconstructed complete analysis of the ore. The serpentine came from Moa Bay. It is of special interest to compare these analyses with a similar analysis from Mayari later given. In quoting Mr. Weld's analyses, the components have been rearranged to correspond with the order now almost universal in rock analyses.

	Serpentine, Moa Bay	Iron Ore, Moa
SiO_2	37.29	1.71
TiO_2	See note	0.14
Al_2O_3	1.33	11.60
Fe_2O_3	....	66.90
Cr_2O_3	See note	2.65
FeO	8.55	....
MnO	tr.	0.80
NiO	See note	0.60
MgO	36.53	See note
CaO	0.29	See note
K_2O	tr.	See note
Na_2O	0.39	See note
H_2O	15.27	13.15
P_2O_5	0.07	0.07
	99.72	97.62

NOTE.—Under the first analysis, 0.28, the balance to 100 per cent., is assigned to TiO_2, Cr_2O_3, and NiO. Under the second analysis, 2.38, the balance to 100 per cent., is ascribed to MgO, CaO, K_2O, and Na_2O.

W. L. Cumings and B. L. Miller specially studied the field north of Camaguey, which their predecessors had apparently named the Cubitas, but which, because of its separated position from the Cubitas Mountains proper, they prefer to call San Felipe. They also add interesting and important details on the Moa field. The San Felipe ores contain in their lower portions impressive amounts of chert, which fails in the other districts. They note large, recognizable crystals of pyroxene in the serpentine, and remark the relations of the serpentine with massive Triassic limestone (p. 119 of their paper) along the Jiguey River. Hayes, Vaughan, and Spencer do not mention Triassic rocks in their stratigraphic column (p. 33 of their report to General Wood) and it would be of great interest to record the evidence. Messrs. Cumings and Miller were well aware that all the iron in the ore was not hydrated, and mention both

magnetite and hematite (p. 128) as components. Small pellets of hematite and some magnetite were detected in microscopic slides of serpentine (p. 135), so that these minerals are regarded as survivors from the serpentine.

C. K. Leith and W. J. Mead visited all three of the large districts but gave special attention to the Mayari. They emphasize the presence of bauxite ($Al_2O_3.2H_2O$) and gibbsite ($Al_2O_3.3H_2O$) in the ore. Eleven

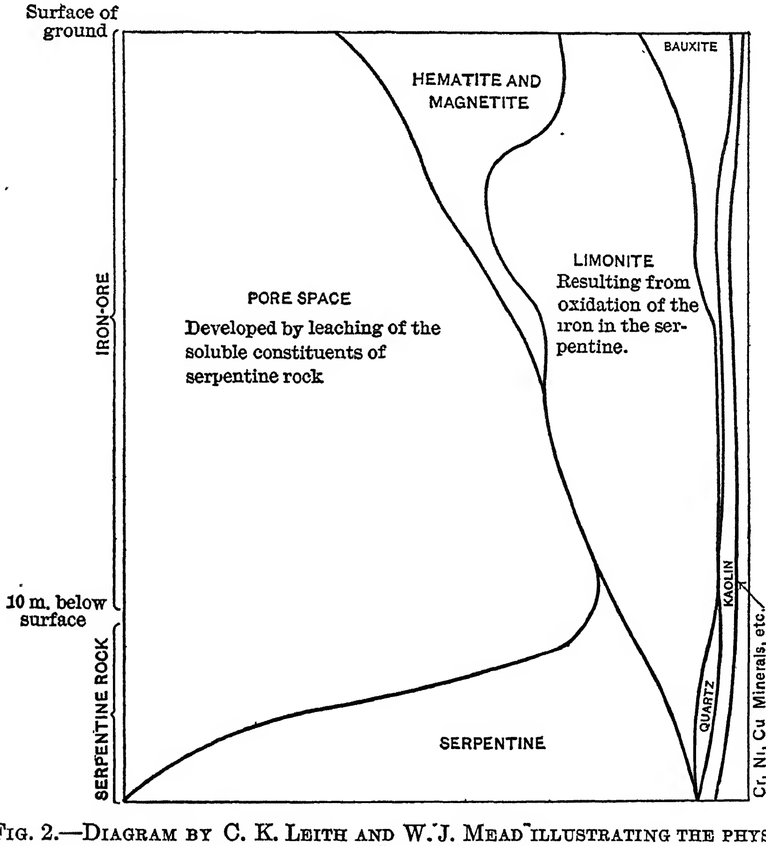


FIG. 2.—DIAGRAM BY C. K. LEITH AND W. J. MEAD ILLUSTRATING THE PHYSICAL AND MINERALOGICAL CHANGES OF THE SERPENTINE IN ITS PASSAGE TO IRON ORE. REPRODUCED FROM P. 93 OF VOL. XLII OF THE *Transactions*.

samples were taken in a 10-m. section of the ore and were analyzed, as was also the bedrock serpentine, so as to trace out the changes. The analyses are not published, but a chart, plotted so as to show the pore space, progressively left by the lost components, and the relative volumes occupied by the residual minerals, is given and is here reproduced (Fig. 2). In order to run some comparisons with the mineralogy of the ore as determined by recasting some new analyses, the writer has scaled off in fiftieths of an inch the cross-section of the chart, first at the top,

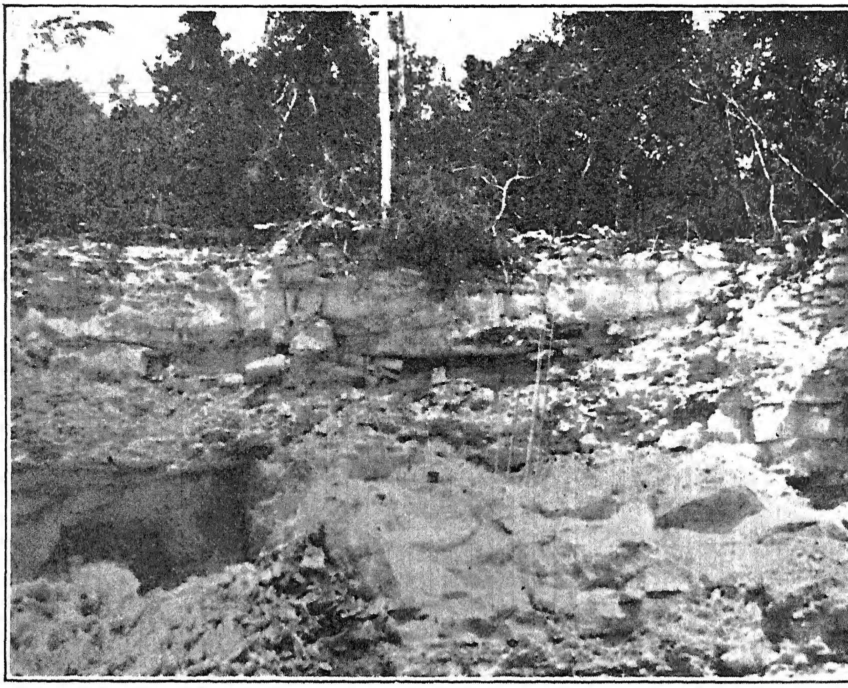


FIG. 3.—LIMESTONE QUARRY AT ARROYO SEBORUCO, 12 MILES FROM FELTON.

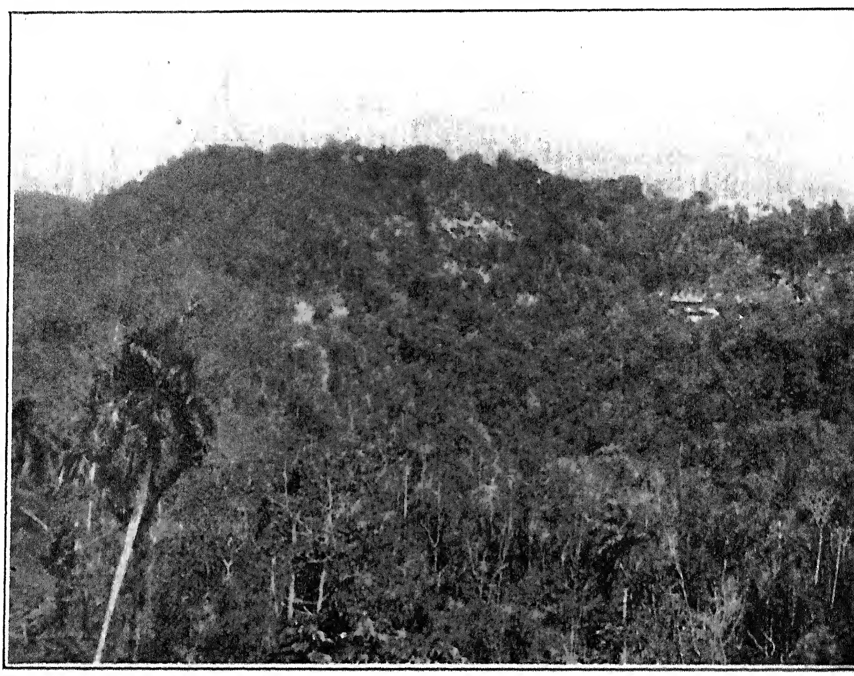


FIG. 4.—VIEW OF LIMESTONE FOOT HILL OF SERPENTINE PLATEAU, LOOKING WEST FROM HEAD OF LOWER INCLINE.

next in the middle of the ore, and lastly at the bottom of the ore. After leaving out the pore space the other proportionate measurements have been multiplied by the respective specific gravities of the minerals involved, and the products have been recalculated to percentages with a total of 100, so as to express the percentages of the minerals by weight. These will be used in comparison later on. The general chemistry and physical changes in the passage from serpentine to ore are carefully reviewed by the two authors. On the geology the following sentences are significant:

"Geologists are substantially agreed that the ores of the Moa and Mayari fields are residual or mantle deposits resulting from surface alterations in place, of serpentine rock, which in turn probably represents the alteration of some other rock like a peridotite not yet disclosed by underground explorations. With the serpentine there are present very minor quantities of intrusive dike rocks, high in alumina, which by the surface alterations yield clay, not iron ore."

This summarized review will afford, the writer hopes, a serviceable idea of what has been already recorded. Some further notes on the general geology and some petrographic details of the serpentine will be next given and then the actual and so far as possible the proportionate mineralogy of the ore will be further discussed.

Geology

At the coast and along the low bluff arising from the water, a marly clay with lumps of limestone intermingled forms the surface. The same general formation extends for some miles back, and appears as thin, flat beds of tender limestone in thicker beds of clay or marl. Twelve miles from the coast, quite solid, white limestone appears in flat beds and is quarried as shown in Fig. 3.

The writer looked over the quarry with care in the search for fossils, having in mind shells or corals of fairly large size. In the brilliant summer sunshine, none were observed on the glaring white ledges. The impression was gained that none were present and the statement to this effect was made in the edition of this paper published in the *Bulletin* of the Institute for February, 1915, p. 136. On closer examination in the laboratory it has been discovered that the limestone itself is made up of minute organisms, which are clearly revealed by microscopic examination of thin sections. The principal component of the rock is a small foraminifer, 0.1 to 0.3 in. (2.5 to 7 mm.), of the genus Orbitoides, presumably an Oligocenic form. Figs. 5 and 6 illustrate it. The accompanying description has been kindly summarized by Marjorie O'Connell, M. A., Curator in Paleontology in Columbia University, who first detected the evidences of organisms in the rock.

FIG. 5.—CROSS-SECTION OF *Orbitoides Kempi*, IN LIMESTONE FROM THE QUARRIES SHOWN IN FIG. 3. ACTUAL FIELD, 2.5 MM. OR 0.1 IN.

FIG. 6.—SECTION SHOWING THE TAPERING EDGE OF *Orbitoides Kempi*, AS WELL AS MINUTE FORMS OF OTHER ORGANISMS. ACTUAL FIELD, 2.5 MM. OR 0.1 IN.

Orbitoides kempi O'CONNELL N. SP.[2]

The foraminiferan which makes up the Seboruco limestone is an Orbitoides of a species new to North America. The only Orbitoides heretofore described from this country is *O. mantelli* from the lower Oligocenic of the Gulf States. This form from Cuba, then, so different in all of its characteristics from the *O. mantelli*, is of great interest as perhaps forming a basis of correlation over wide areas. The most closely related species is *O. dispansa* Sowerby, which occurs in the Nummulite limestone of the Eocenic of the Bavarian Alps, in the Priabona beds of the lower Oligocenic of Massano, and elsewhere in Italy, as described by Gümbel.[3] The age of the limestone in Cuba would thus seem to be upper Eocenic or lower Oligocenic, probably the latter.

The fossil has the form of a sphere surrounded by a wavy rim of horizontal extent, the whole resembling two flat-brimmed, round-crowned hats placed base to base. The total extent of the shell ranges up to 7+ mm., while the thickness of the central sphere is 2.5 mm. in one of the largest individuals. Like all of the other species of Orbitoides, this form begins with a single layer of chambers which multiply in one plane and assume a circular outline; but it is in the subsequent growth that the distinguishing specific characters appear, for the chambers at the central portion of the shell increase more rapidly than do those near the periphery and in consequence a swelling is produced which finally gives the spherical form which is so characteristic. The rim, on the other hand, shows little chamber-increase and remains thin. This is clearly shown in the sections figured, where the central row of larger chambers forms the equatorial zone. Radiating from this row, in section, there appear a number of solid cones alternating with broad reticulated ones. The solid cones mark the more dense structure forming the walls surrounding the reticulated or chambered cones. These appear upon the surface as distinct pits, whereas the solid part appears as raised walls.

Presumably the same formation extends to the plateau and appears as far up as the head of the lower incline. It could also be seen across a valley, as shown in Fig. 4, a photograph taken from the head of the lower incline looking westward. The point of view was at 680 ft. elevation above the sea. Above this point serpentine soon appears and, with the associated dikes of diorite as later described, constitutes the plateau so far as known. Undoubtedly more detailed exploration of the valleys, and of the lower contact of igneous rocks on limestones would bring to light additional facts of much interest. There must be a vertical thickness of exposed serpentine of at least 1,200 ft.

The cuts of the upper incline give an excellent section of the serpentine, since they have been sunk as much as 40 ft. below the surface. Good, fresh rock has thus been afforded for study. Other cuts along the railway between the head of the incline and the station at Woodfred give good exposures, and a third cut made in the bedrock near the southwest workings, known as the Three-Hill mine, has opened up excel-

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[2] Named in honor of the discoverer of the fossil, Prof. James F. Kemp of Columbia University. A complete description of this fossil will be published elsewhere.

[3] C. W. Gümbel: Beiträge zur Foraminiferenfauna der Nordalpinen Eocängebilde, *Abhandlungen der Math.-Phys. Classe der königlich-bayerischen Akademie der Wissenschaften*, vol. x, pt. 1, p. 701 (1866).

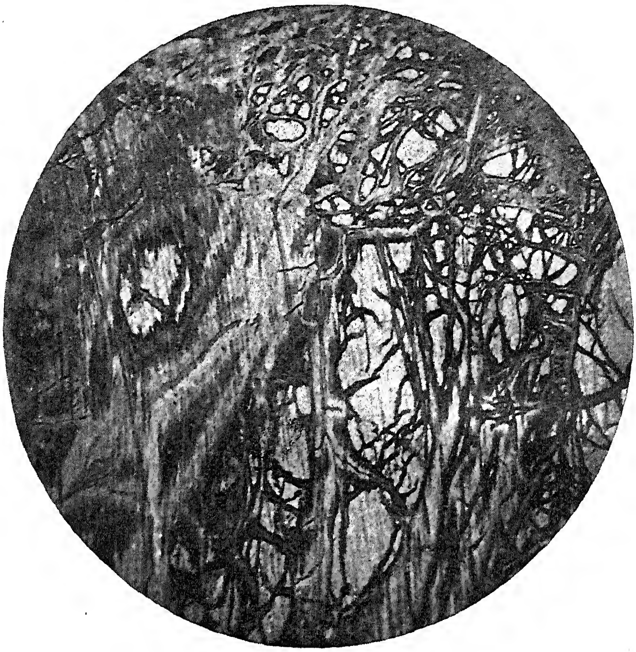

FIG. 7.—LOWER TWO-THIRDS, BASTITE (HIGH RELIEF) AND SERPENTINE (LOW RELIEF) FROM PYROXENE; UPPER THIRD, OLIVINE CHANGING TO SERPENTINE. ACTUAL FIELD, 0.1 IN.; WHITE LIGHT.

FIG. 8.—THE SAME AS FIG. 7, BUT TAKEN WITH CROSSED NICOLS.

lent material for microscopic study. A very fresh and surprisingly unaltered boulder of rock was met in the mines at the Y, where the spur to the northeast branched from the main line, in 1914.

The serpentine is derived from a peridotite which consisted of olivine, and pyroxene, apparently an orthorhombic variety which is now thoroughly changed to bastite and fibrous serpentine. The original rock probably also contained monoclinic pyroxene, since in the slides some extinctions not parallel with the cleavage would suggest it. The chemical change has been so thorough as to destroy most of the optical properties of the original. The bastite has feeble double refraction, giving

FIG. 9.—BASTITE PRESERVING CLEAVAGES OF A PYROXENE CRYSTAL IN RIGHT CENTER. ABOVE IT A DARK PICOTITE. GENERAL MASS SERPENTINE FROM OLIVINE. ACTUAL FIELD, 0.1 IN.; WHITE LIGHT.

pale grays. It still displays prismatic cleavage which has parallel extinction. It passes into fibrous serpentine, as is illustrated in the lower two-thirds of Fig. 7. The serpentine fibers maintain parallelism with the prismatic cleavage of the original pyroxene. The fibers are curved slightly in Fig. 7, suggesting some pressure effects. In the change to serpentine some highly refracting mineral has developed in the cracks, which behaves like quartz, but extinguishes at various angles across the elongation. It is shown in Fig. 8, which is the same field as Fig. 7. In Fig. 9 is a pyroxene crystal with its prismatic cleavages well shown and in addition pronounced partings parallel with the vertical pinacoids.

The dark, angular grain just above the pyroxene crystal is a brown, isotropic mineral of high index and is probably the chrome spinel, picotite. It is quite frequently seen in the slides and is believed to be picotite because the dark brown mineral becomes transparent too readily to be chromite. The slides show occasional opaque metallic grains, which are doubtless magnetite and chromite. The serpentine develops in the olivine in cross fibers, along cracks, and is well illustrated in Fig. 10. A further stage, from which the olivine has almost all disappeared and serpentine with the so-called mesh structure has taken its place, is illustrated in Fig. 11. Numerous black specks of secondary iron ore are

Fig. 10.—Showing general derivation of Serpentine from Olivine. Actual field, 0.1 in.; white light.

shown. Finally, in Fig. 12, we have only serpentine left, with the mesh structure emphasized by taking the photograph with crossed nicols.

In the cuts in the serpentine along the upper incline rather poorly defined streaks of dense black or dark-gray rock can be frequently detected. So much movement or crushing has taken place, either from dynamic disturbance or from the swelling and adjustment incident to the change from peridotite to serpentine, that their regularity is greatly obscured. Doubtless they were once dikes, but now on the edges they shade into the schistose serpentine and probably have themselves altered to it in part. Under the microscope they are all diorites with strong suggestions of original diabasic texture. The long banded rods in Fig.

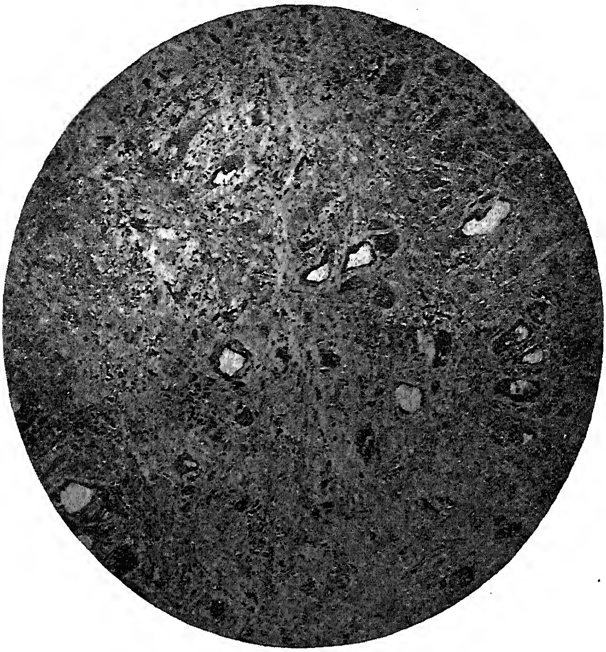

Fig. 11.—Showing almost complete change of Silicates to Serpentine. Actual field, 0.1 in.; white light.

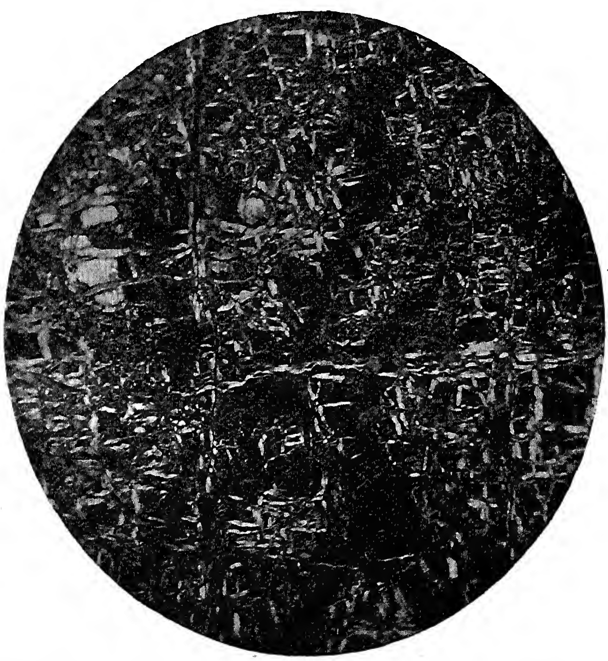

Fig. 12.—Similar view to Fig. 11, but taken with crossed nicols to bring out the Mesh Structure of the Serpentine.

13 are the plagioclase. The other minerals of uniform gray to black tints are hornblende. Magnetite in small grains can also be detected in the slides. In other dikes the hornblende becomes more abundant and the plagioclase less. In Fig. 14, taken in white light, the darker mineral making up three-quarters of the field is all hornblende. The lighter areas are plagioclase. Some dead-black magnetite is also shown. The boulder mentioned above as found near the Y of the mine railway is a coarser diorite, rich in feldspar, which even shows micropegmatitic structure. Although appearing as a loose boulder, this rock proved to be surprisingly unaltered in the thin section, all the minerals being clear

FIG. 13.—DIORITE WITH DIABASIC TEXTURE. THE DARK AND LIGHT BANDED MINERAL IS PLAGIOCLASE; THE DARK MINERAL IS HORNBLENDE. CROSSED NICOLS. ACTUAL FIELD, 0.1 IN.

and fresh. It must indicate a dioritic dike in the serpentine. Mr. Spencer has reported diabase dikes to be numerous and widespread in the serpentine, and, as he crossed this area years before mining was begun, he must have observed them along the steep sides of the plateau, since practically no outcrops appear on top. One cannot but suspect from the textures of the dikes here illustrated that the hornblende is secondary after augite, and the slides justify this surmise. A few cases of unchanged augite have been observed in the midst of the hornblende, whose leek-green color would almost of itself be demonstrative of a secondary variety. In slides made from several dikes gathered on the

incline the feldspar has passed to an aggregate of not strongly bi-refracting rods, which are probably some zeolite. Doubtless this is the first stage in the alteration, which in the long run yields laterite. Although no chemical analyses have been made, all these diorites will run much higher in alumina than the serpentine. They will certainly yield from 15 to 18 per cent., and are doubtless responsible for some of the alumina in the ore and its local exceptional abundance.

An analysis of the fresh serpentine from the Three Hill mine cut, from which Figs. 7, 8, and 9 were taken, has been kindly made by T. C. Kraemer, chemist of the Spanish-American Iron Co. at Felton. By its side is placed the analysis given by C. M. Weld, as earlier noted:

	I Serpentine, Woodfred	II Serpentine, Moa
SiO_2	37.28	37.29
Al_2O_3	2.45	1.33
Fe_2O_3	5.14	....
Cr_2O_3	1.68	....
FeO	5.14	8.55
MnO	....	tr.
NiO	0.30	....
MgO	34.59	36.53
CaO	none	0.29
K_2O	....	tr.
Na_2O	....	0.39
H_2O (combined)	12.80	15.27
H_2O (absorbed)	0.91	....
P_2O_5	0.013	0.07
	100.303	99.72

In analysis II, Mr. Weld assigns to the difference between the total and 100, TiO_2, Cr_2O_3, and NiO, amounting to 0.28 per cent.

Much interest attaches to the recasting of these analyses into the percentage composition of the component minerals of the rocks. At first sight the presence of so much Fe_2O_3 in the analysis of the Woodfred sample strikes one as unusual, since in general in serpentine the iron oxide is chiefly in the state of protoxide. If, however, in recasting we assume it to be ferrous, we have far too little silica to satisfy the serpentine molecule $3(MgO.FeO)$, $2H_2O$, $2SiO_2$. The ferric oxide has therefore been arbitrarily distributed between limonite and magnetite, the governing motive being the exhaustion of the silica. As checked by study of thin sections, we are thus not very far from the true components of the rock or their relative amounts. The chief variation lies in the undoubted presence of some unaltered olivine. Bastite is generally credited with the same composition as serpentine. Mr. Weld's analysis is stated to be the average of a number of samples. When one attempts to recast it, assigning the magnesia and the ferrous iron to the serpentine molecule, there is lack of silica, and the difficulties are increased by assuming any

probable silicate for the soda and lime. Any one of the averaged individual analyses would be better for recasting than the average of them all.

Fully realizing, therefore, that the percentages given by the recasting of the Woodfred sample are approximations rather than mathematically correct expressions, we have some idea of the minerals or compounds whose alteration has produced the ore. Sharper definition is thus given to our conceptions of the necessary changes than if we had only the analysis before us. The total does not exactly check with the analysis, from small rejections of decimals, here and there.

FIG. 14.—MORE BASIC DIORITE WITH PREDOMINATING HORNBLENDE. ACTUAL FIELD, 0.1 IN.; WHITE LIGHT.

	Serpentine, Woodfred	
Magnesia serpentine.....	79.33	Total serpentine 85.986
Iron serpentine.........	6.174	
Nickel serpentine........	0.482	
Kaolinite...............	·1.548	
Bauxite................	2.346	
Limonite...............	4.114	
Magnetite..............	2.320	
Chromite...............	2.472	
Unassigned water.......	0.396	
Absorbed water.........	0.910	
P_2O_5..................	0.013	
	100.105	

The change to ore obviously involves the entire disappearance of the magnesia; the nearly total removal of the silica; the probable change of the nickel-serpentine to one or more of the other hydrated silicates such as have been named from various nickel mines—genthite, connarite, garnierite, etc., and of which F. W. Clarke[4] states: "These silicates are rarely if ever found as definite mineral species, although they have been described as such." One called nepouite has, however, the same formula as nickel-serpentine, $3(NiMg)O, 2SiO_2.2H_2O$, or $(NiMg)_3Si_2O_7, 2H_2O$ as cited by Dr. Clarke. In the passage to ore, whose composition is later shown, some nickel surely migrates downward, because it is not increased in percentage in the same ratio as the alumina, and the serpentine below the ore is much richer in nickel than the sample above analyzed. Instead of 0.3 per cent. we may find 2 per cent. and over of NiO. The kaolinite breaks up in large degree to bauxite ($Al_2O_3.2H_2O$) or some other aluminum hydrate such as gibbsite (called also hydrargillite), $Al_2O_3.3H_2O$. There seems to be no good reason why there should not be present also $2Al_2O_3.3H_2O$, corresponding with limonite, but no such mineral has yet been named. Even bauxite itself may be interpreted as an intimate mixture of diaspore, $Al_2O_3.H_2O$, and gibbsite, as Dr. F. W. Clarke remarks (p. 472), since the last two are known in a crystallized condition while bauxite is not. The chromite and magnetite survive with little change, both being very resistant minerals. Hematite may also be a mineral in the serpentine, as considered by Cumings and Miller, even though not calculated as such in the above recasting. It may survive in the residual product. The iron passes into some form of hydrated oxide, as will be later shown. Limonite, $2Fe_2O_3.3H_2O$, probably is in excess over goethite, $Fe_2O_3.H_2O$, and turgite, $2Fe_2O_3.H_2O$. Some iron doubtless has also escaped. All these changes are facilitated by the situation of the serpentine on a high plateau, whose top is 1,500 to 1,900 ft. above the surrounding lower country, and which has extremely steep sides. Underground discharge for the percolating rain water has been extremely free, since serpentine is pre-eminent among rocks for checks, slips, and cracks. The great increase in volume incident to the combination of so much water with the anhydrous silicates of the peridotite makes readjustments on a grand scale inevitable.

The Ore

In the faces of the pits as now extensively exposed, the observer can readily note that there are three distinct layers: An upper, of crimson-brown hue; a middle one, yellowish brown, and a bottom layer of a lighter shade of yellowish brown. In one face, called the Three-Hill cut, the

[4] F. W. Clarke: Data of Geochemistry, *Bulletin No.* 491, *U. S. Geological Survey*, p. 664 (1911).

upper was noted at 5 to 6 ft.; the middle at 6 to 12 ft.; and the lower at 4 to 6 ft. The two upper layers are illustrated in Fig. 15. Analyses of each layer, but from another station, will be subsequently given. As distinct from these varieties the engineers in charge have noted that in the occasional and rather rare spots in the residual mantle where the iron percentage is too low for mining there appears at the surface a peculiar purple color, quite easily recognized and an indication of high alumina in the samples. The color appears to be due to the relatively rich admixture of normally white bauxite with the darker-hued brown iron ore. The explanation of the higher alumina is to be found in local changes in the original rock, as later set forth.

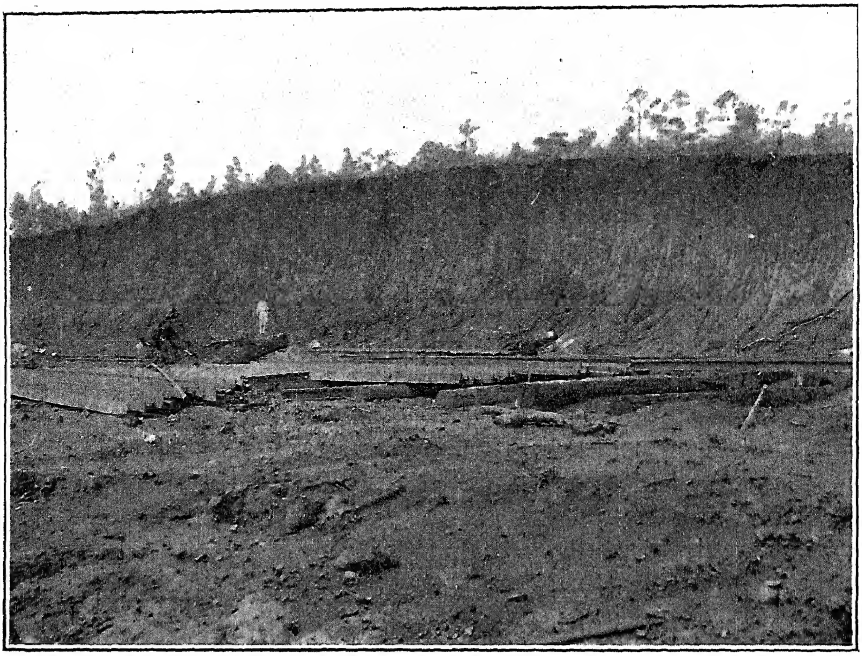

Fig. 15.—Face left by Steam Shovels in Three-Hill Mine, and showing top, dark layer, and middle, lighter layer. The top dark layer has caved down on the base, concealing bottom layer.

In some places, at the surface or a few feet below it, slabs and even continuous sheets of solid iron hydrate appear and afford the cellular varieties of brown ore, very similar to the crusts and lumps long familiar in the mines of the Appalachian belt. As earlier stated, the solid ore is called plancha. In the residual mantle shots and larger lumps of solid brown ore are at times intermingled, chiefly in the upper, darker layer. The general run of the ore is, however, earthy, and reminiscent in the strongest degree, alike in color and texture, of the Mesabi ores. The higher con-

tent of absorbed water, the higher alumina, and the lower silica of the Mayari ores give them perhaps a somewhat more spongy aspect than one notes on the Mesabi range; on the other hand, at Woodfred there is no overburden whatsoever, and the ore is obtained from grassroots to bedrock.

The recently mined ore has a peculiar mealy character, reminding one of nothing so much as dampened meal, but as it dries out this character disappears. No doubt the colloid nature of the hydrates of alumina and iron is the cause of the peculiarity.

The following analyses made by T. C. Kraemer, chemist of the Spanish-American Iron Co., were based upon samples gathered by the writer to illustrate the three contrasted layers. The samples were taken as nearly as practicable in a vertical section.

	I Surface Ore. Sta. 11,35,10. Crimson-brown	II Middle Layer. Sta. 11,35,10+15 Yellow	III Bottom Layer. Sta. 11,35,11 Yellow
SiO_2	2.26	2.70	7.54
Al_2O_3	14.90	7.13	4.97
Fe_2O_3	68.75	71.89	64.81
Cr_2O_3	1.89	3.17	3.66
FeO	0.77	1.29	1.49
NiO	0.74	1.60	2.75
MgO	....	....	1.50
H_2O combined	11.15	12.90	12.75
Total	100.46	100.68	99.47
Metallic iron	48.65	51.32	46.52
Metallic nickel	0.59	1.20	2.10
H_2O absorbed[a]	4.62	9.72	27.00

[a] Determinations of absorbed water based on original sample. Other determinations on dried sample at 110° C.

From these analyses we draw the following conclusions:

Silica progressively increases from above downward. It is relatively high in III because of bits of included serpentine and probably because of the descent of the silica in solution and loss in largest proportion from the uppermost layer, which has been longest exposed to alteration.

Alumina decreases downward because it is the least soluble of all the components and therefore reaches a maximum in the most weathered portion.

Ferric oxide, as we pass downward, increases and then declines. In III the silica and magnesia of the bits of serpentine relatively reduce it. Doubtless, in the long run descending rain waters dissolve some iron from the upper layer and re-deposit it in part in the yellow middle layer. Some probably runs off altogether.

Chromic oxide progressively increases downward, a relation at first causing surprise. Since chromite is an extremely resistant mineral,

one would naturally expect it to become concentrated near the top. It must either work its way downward because of its relatively high specific gravity or else it must appreciably yield to waters in the long run. The former supposition is more probable.

Ferrous oxide impresses one at first sight as being sympathetic with chromic oxide, although a little below the requirements of chromite. The difference could be explained by small percentages of magnesia or even of nickel oxide in the chromite. Tests with the magnet, however, prove the presence of appreciable quantities of magnetite. It therefore seems more reasonable to assign the FeO to magnetite; and to infer from the insoluble nature of chromite that its FeO was finally determined as Fe_2O_3. In recasting the analyses all the FeO has been used for magnetite, and enough Fe_2O_3 to furnish the necessary FeO for chromite has been taken for this purpose.

Nickel oxide progressively increases downward. It may be in very small part in the chromite. It is undoubtedly present as some hydrated nickel silicate. It is probably affected by the rain waters and has been removed by them to a relatively great degree from the upper layers. Concentration in the bedrock and in the bottom layers of ore is quite certainly shown by analyses, additional to the three cited.

Magnesia practically disappears with the general breaking down of the serpentine.

Combined water shows a slight increase, because surviving serpentine in the bottom layer accounts for some, and increased percentages of iron, presumably in the molecule, $2Fe_2O_3.3H_2O$, in the middle layer for the rest. There is reason to infer the increased percentage of magnetite and perhaps hematite in the top layer. Leith and Mead plot these two as forming more than half the solids by volume, and this would involve an even greater percentage by weight, because the minerals bauxite and kaolinite of low specific gravity enter in an important way into the residue. The writer's tests with the magnet and by microscopic study of the samples gathered for the analyses here published showed that the shots of ore were magnetic to an appreciable degree, perhaps 5 to 10 per cent. of them, but the fine ore was practically inert. The shots were all discolored with brown ore and when crushed showed a goodly portion of limonite in the powder.

If we seek to combine the iron and all the water of I into the common limonite molecule, there is too little water; in II there is a slight excess of water; and in III there is a large excess. But the alumina in each calls for some water, and it is therefore clear that all the iron is not in the limonite molecule, $2Fe_2O_3.3H_2O$, but there must also be present some iron hydrate calling for less water. From microscopic study, we are certain that there is a little magnetite, and this helps to solve the difficulty, but there must also be present large proportions of goethite, $Fe_2O_3.H_2O$, and

there may be some turgite, $2Fe_2O_3.H_2O$. The absorbed water is exceptionally low in I and II, and the small percentage is undoubtedly due to evaporation during a period of slight rainfall.

Much interest attaches to the recasting of these analyses in the endeavor to bring out the actual or probable proportions in which the constituent minerals enter and the identity of the minerals themselves. The low percentage of silica makes clear that kaolinite, $2H_2O.Al_2O_3.2SiO_2$, one of the first minerals to be considered, can be present in only small amounts. The presence of chromic oxide makes justifiable and indeed unavoidable the conclusion that chromite, $FeO.Cr_2O_3$, is also present. Recasting shows that there is from a little to a good deal too much Cr_2O_3 in the analyses for the reported FeO. As stated above, we infer from this, since chromite is a very insoluble mineral, that the reported FeO is in magnetite. On this assumption the magnetite has been calculated, assigning to it all the reported FeO, and then taking from the Fe_2O_3 enough to correspond to the FeO required for chromite. The resulting percentages are so close to the general indications of microscopic study as to be worthy of much confidence. We infer that the greater part of the nickel is combined with silica and water as some form of hydrated silicate. The normal nickel-serpentine molecule has been used. Leaving one side the nickel as one of the minor components, we recognize the fact that the iron hydrates and the alumina hydrates are the large components.

As a further preliminary to the results of attempted recasting, a brief summary may be given of a careful microscopic examination made by Dr. Charles P. Berkey for the Spanish-American Iron Co., in order to determine the condition of the iron minerals and their relations to other components. The ore proved to be in largest part an extremely intimate mixture of some iron hydrate, assumed to be limonite; and some alumina hydrate, assumed to be bauxite; with minor proportions of recognizable chromite and admixtures in minute particles of quartz, epidote, hornblende, and possibly feldspar. By fractional treatment with hydrochloric acid, and study of residues, the above conclusions were reached. The sample was different from the ones used in the three analyses here published, but the results give us a definite and interesting point of view. Chamosite and thuringite, the hydrated silicates of iron, familiar in the minette ores of Luxemburg and Lorraine, were considered, but no reason appeared for inferring their presence.

In the tables of recast results, different assumptions are made and tabulated for iron and alumina hydrates in the endeavor to work out combinations of the water, which is too small in amount for limonite, $2Fe_2O_3.3H_2O$, and bauxite, $Al_2O_3.2H_2O$, alone. The methods employed are those now widely used for rock analyses.[5]

[5] Kemp: *Handbook of Rocks*, pp. 161 to 179. Cross and others: *The Quantitative System for the Classification of Igneous Rocks* (Chicago, 1903). Finlay, G. I.: *Igneous Rocks* (New York, 1913).

	I Top Layer, Recast			II Middle Layer	III Bottom Layer
	Ia	Ib	Ic		
Limonite	39.66	21.32	....	59.09	69.96
Hematite	31.68	....	....	....	....
Turgite	....	50.02	30.76	....	....
Goethite	....	....	40.58	18.87	....
Magnetite	2.55	2.55	2.55	4.18	4.87
Chromite	2.79	2.79	2.79	4.85	5.58
Bauxite	17.98	17.98	17.98	7.45	1.34
Kaolinite	4.05	4.05	4.05	3.78	9.80
Ni-serpentine	1.30	1.30	1.30	2.69	4.70
Mg-serpentine	....	....	....	....	3.45
Excess water	....	....	....	....	0.04
	100.01	100.01	100.01	100.91	99.74

In recasting analysis I, the relatively small amount of water has compelled three assumptions. In the first, after providing for bauxite, kaolinite, and nickel-serpentine, all the water is assigned to the limonite molecule and the excess of ferric iron is calculated as hematite. So much hematite is, however, contradicted by microscopic study of the sample. The magnetite and chromite account for approximately as much as one actually observes of the iron minerals which do not afford on crushing a yellow or brown powder. Therefore, a second recasting was carried out by allotting the water to limonite, $2Fe_2O_3.3H_2O$, and turgite, $2Fe_2O_3.H_2O$. By this assumption yellow-brown hydrates of iron alone are used. The third recasting was made also to use only iron hydrates; but assuming turgite, $2Fe_2O_3.H_2O$, and goethite, $Fe_2O_3.H_2O$, one sees that hematite is not a necessity in any great quantity. In the diagram given by Professors Leith and Mead, if we select sections at the top, middle, and near the bottom of the ore, scale off a proportionate part for each mineral by its specific gravity, and then reduce the products to percentages, we will turn the diagram into percentages by weight. In doing this the following specific gravities have been used: Magnetite and hematite, 5; limonite, 4; bauxite, 2.4 (as given by J. C. Branner, *Journal of Geology*, vol. v, p. 270, 1897); kaolinite, 2.5; chromium, nickel, and copper minerals, 4.5 (a value probably high); serpentine, 2.6.

	I Top	II Middle	III Bottom
Hematite and magnetite	64.8	15.0	....
Limonite	19.2	72.4	75.4
Bauxite	12.7	2.4	....
Kaolinite	1.6	3.0	8.4
Chromite, etc	1.7	7.2	9.0
Serpentine	.....	......	7.2
	100.0	100.0	100.0

In the smaller components, kaolinite, chromite, and serpentine, the results are generally reasonably parallel with those obtained by recasting the three analyses given above. The bauxite is not beyond the allowable differences of samples. The chief contrasts are in the distribution of the iron oxides among the several iron minerals in the top layer. The samples used in the present paper indicate less hematite and magnetite.

Samples were also collected at station 21.02.16 of the lean so-called ore, of purple color, and too low in iron for mining. A second sample of a cellular decomposed mass, supposed at the time to be bedrock, was taken from the base of the exposed section. Both were kindly analyzed by T. C. Kraemer. Apparently the samples were derived from the weathering of the peridotite, where penetrated by diorite dikes.

	I Upper Portion	II Lower Portion
SiO_2	6.88	2.28
Al_2O_3	23.44	39.80
Fe_2O_3	54.46	34 44
Cr_2O_3	1.89	0.27
FeO	......	......
NiO	0.25	0.09
H_2O combined	13.84	24.01
	100.76	100.89
H_2O absorbed	4.72	1.86

	I Upper Portion	II Lower Portion
Limonite	38.17	40.06
Hematite	20.80	......
Bauxite	24.02	19.05
Gibbsite	......	36.19
Kaolinite	14.49	4.90
Ni-serpentine	0.41	......
Chromite	2.85	0.43
NiO not used	......	0.09
	100.74	100.72

In No. I we are forced to use hematite, or appeal again to goethite and turgite, which latter step in this case has not been taken. In No. II the water is so high as to necessitate the presence of gibbsite, $Al_2O_3.3H_2O$, along with bauxite. The gibbsite molecule is important in the laterites. Undoubtedly in all these processes of weathering colloids are formed, since, with the exception of chromite, hematite, and in the previous analyses magnetite, practically all the molecules employed appear in amorphous and structureless forms, which are most reasonably explained by assuming an original colloid. At the International Geological Congress, held in Toronto, in August, 1913, Prof. Paul Krusch, of Berlin, emphasized the importance of this form of matter both in the results of surface weath-

ering and in a few possible situations placed deeper within the earth.[6] The hydrated nickel silicates received special mention and were interpreted as colloidal precipitates.

Recently Described Areas of Laterite Ores

In the earlier papers cited on the Mayari and related districts mention is made of similar ores at Staten Island, N. Y.,[7] and Clealum, Wash., and of others in the Mediterranean region. Besides these two, additional instances have recently been described to which reference may be made in conclusion. Prof. Alfred Lacroix, of Paris, has been for over 15 years carrying on extended studies of the tropical alteration of rocks in French Guinea in latitude 10° N.[8] Professor Lacroix defines laterites (p. 259) as "The products of the decomposition of all rocks containing silicates of alumina, and which, from the chemical point of view, are characterized by the predominance of the hydroxides of alumina and iron generally with the oxide of titanium, after the elimination more or less complete of the other elements of the fresh rock: alkalies, lime, magnesia, and silica." While the original peridotite of the Mayari iron ore is one of the poorest in alumina of all igneous rocks, yet there was enough in it so that the residual products of its decay became relatively high in this oxide. Professor Lacroix gives (p. 296) the accompanying analyses, which range from much less altered peridotite than we have yet obtained at Mayari to iron ore apparently like the crusts of Cuban plancha.

Analysis (d) is analogous to the Mayari ore and therefore much interest attaches to the following comment of Professor Lacroix, p. 297:

> "As for analysis (d) it reveals a complete disappearance of the silica. The interpretation of the constituents of the hydrates is uncertain. The red coloration, of variable shades, in the specimens studied leaves no doubt about variations in the hydration of the iron. If we assume that the hydrate of alumina is the one with three molecules of H_2O, the remainder of the water, which is combined in the hydroxide of iron, makes necessary for the latter a composition less hydrated than that of goethite. If on the contrary, we assume a hydrate of alumina with one molecule of water, then the hydroxide of iron would have the composition of goethite; but any such hypothesis ought to be rejected, since this mineral certainly does not exist in this laterite. Microscopic examination only proves the presence of a mixture of hematite and limonite."

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[6] Paul Krusch: Primaere und sekundaere Erze unter besondere Beruecksichtigung der "Gel" und der "schwermetallreichen" Erze (Primary and Secondary Ores with Especial Consideration of the Colloid Ores and of the Ores Rich in the Heavy metals), *Compte Rendu de la XII Congrès Internationale Géologique*, pp. 275 to 286.

[7] A recent paper by C. R. Fettke, Limonite Deposits of Staten Island, N. Y. (*School of Mines Quarterly*, vol. xxxiii, pp. 1 to 10, July, 1912), interprets the ores in accord with the most recent views.

[8] Les Laterites de la Guinée, *Nouvelles Archives du Museum National d'Histoire Naturelle*, vol. v, pp. 255 to 356 (1913).

	Peridotite a	Peridotite b	Laterite c	Iron Capping d	Iron Ore e
SiO_2	40.01	38.32	12.67	....	2.8
Al_2O_3	2.54	2.66	12.59	4.80	8.7
Cr_2O_3	0.16	0.16	....	0.20	tr.
Fe_2O_3	1.00	4.35	46.84	83.50	77.2
FeO	11.70	11.78	....	....	....
MgO	39.90	36.22	1.26	....	....
CaO	1.68	2.74	0.04	....	....
Na_2O	1.07	0.16	....	....	....
K_2O	0.52	0.06	....	....	....
TiO_2	....	0.28	0.55	....	....
H_2O combined	1.10	3.38	15.32	10.18	11.4
Insoluble	....	....	10.73[a]	1.70[a]	....
	99.68	100.11	100.00	100.48	99.6

[a] The insoluble matter contains the chromic oxide in picotite.

In the Mayari ore, scales or needles which would correspond to the typical forms of goethite have not been observed, but as goethite is also believed to be at times massive or apparently amorphous the writer did not feel sure of its absence in the earthy and indefinitely flocculent mixtures to be seen in the ore when puddled with water under the microscope. Calculations based on this molecule therefore did not seem beyond the range of possibility. Professor Lacroix uses the special name of stilpnosiderite (an old term for a variety of limonite) for the colloid having the composition of limonite. Turgite, however, he concludes is a mixture of hematite and stilpnosiderite, such that the formula $2Fe_2O_3.H_2O$ is obtained. If a mixture corresponding to four hematite molecules and one limonite chanced to be analyzed, the formula of turgite would result. In French Guinea the same darkening of color is noted in the surface layer, and it is explained by Professor Lacroix as due to the dehydration of the limonite (stilpnosiderite) by the sun. The only two demonstrated hydrates of alumina in nature are believed by Professor Lacroix to be diaspore, $Al_2O_3.H_2O$, and gibbsite (or hydrargillite), $Al_2O_3.3H_2O$. Both are known crystallized, but both appear to form colloids as well. Bauxite is not believed to be of definite composition, but to be a mixture of these two. So far as study of the Mayari ore has gone, no crystallized diaspore or gibbsite has been seen. We therefore appear to be dealing with a mixture of colloids and no inaccuracy of serious moment was involved in using the bauxite formula, $Al_2O_3.2SiO_2$, which is practically a combination of one diaspore and one gibbsite. With superabundant combined water, an excess of gibbsite would be inferred, as in analysis II of the lean ore above. An extended literature has developed in recent years regarding the aluminum hydrates, but such new contribution to general knowl-

edge as it contains is chiefly based on the conceptions of colloids. The papers are, however, hardly germane for further citation here.

A very recently discovered body of ore, similar in all respects to the Mayari deposits, has been reported from the northern portion of the island of Mindanao in the Philippines. Its nature seems to have been first recognized by H. F. Cameron[9] of the Government Engineers' staff, who had had earlier experience in the Mayari district in Cuba. The ore is a surface mantle produced by the tropical weathering of basic, igneous rocks in latitude 9° N. Reserves of 800,000,000 tons are estimated and the following analysis is given:

SiO_2	Al_2O_3	Cr_2O_3	Fe_2O_3	NiO	H_2O	S	P	Total
1.20	12.20	1.28[a]	77.71[b]	none	7.63	tr.	tr.	100.02

[a] 1.28 Cr_2O_3 = 0.88 Cr. [b] 77.7 Fe_2O_3 = 54.4 Fe.

This one sample is richer than the run of the Mayari ores by the year (48 to 49 Fe), but one would need to be assured by extensive boring and sampling that so high a yield as 54.4 could be maintained for long periods.

The thanks of the writer are due in full measure to Charles F. Rand, President of the Spanish-American Iron Co., for every facility to study the ores. Acknowledgments should also be made to the officers of the company at Felton and Woodfred for many courtesies.

[9] Personal letter to the writer.

The Boulder Batholith of Montana[1]

BY PAUL BILLINGSLEY, BUTTE, MONT.

(New York Meeting, February, 1915)

THE term Boulder batholith was first applied in 1897 by W. H. Weed[2] to the extensive mass of granite in western Montana within whose borders occur the ore deposits of Butte. In a general way this was known to extend from Helena on the north to the Highland Mountains near Butte on the south, and to be relatively narrow in its east-west diameter. Its intrusive relations to the surrounding sedimentary rocks, and its probable Tertiary age, were described by Weed, but in many ways his statements, based upon incomplete data, have required revision. Most notable of the errors of this description of the batholith were the failure,[3] since corrected, to recognize the intrusive contact of the granite with the capping of andesite, and the complete absence of any reference to the many extensions of the granite beyond the somewhat narrow limits of its largest exposure.

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[1] The field observations upon which this paper is based have been made during the past five years by J. A. Grimes, of Butte, and the author. Over 300 thin sections, prepared and examined by A. H. Smith, of Gateway, B. C., have assisted in determining doubtful points. Reports on mines by Reno H. Sales, F. A. Linforth, and Murl Gidel, of Butte, have been freely drawn upon.

The accompanying geologic map is based primarily upon original field work. A considerable area thus covered has since been mapped in publications of the U. S. Geological Survey, and these maps, where of greater accuracy, have been incorporated. A map of the Helena region accompanying *Professional Paper No.* 74 (Ore Deposits of the Butte District, by W. H. Weed) has required revision, but the more recent map of Knopf covering the same area has been drawn upon freely. Stone's map of the Elkhorn Mountains has likewise helped to fill a gap, while the wonderfully accurate map accompanying *Professional Paper No.* 78 (Philipsburg Quadrangle, by W. H. Emmons and F. C. Calkins) has been used without modification. The only criticism possible on this map is the employment of divers and contrasting colors in representing granitic rocks of common age and origin and with but slight textural and mineralogical differences.

[2] *Butte Special Folio, U. S. Geological Survey* (1897). Also, Granite Rocks of Butte, Mont., and Vicinity, *Journal of Geology,* vol. vii, No. 8, p. 737 (Nov.–Dec., 1899).

[3] *Butte Special Folio,* p 1 (1897).

These outlying areas of the granite, however, did not escape the attention of other and earlier observers, and in the geological reports of the Hayden expeditions,[4] as well as the descriptions of the early quartz mines of the State compiled by R. W. Raymond,[5] references to the granite abound. A common error among these early writers was the general grouping of the granite with the gneiss and schist frequently found, as a complex basal system of Archean age, and for this reason their observations, while remarkably accurate in detail, have failed to bring into prominence the widespread extent of the intrusive granite. Absence of detailed study of the gaps which separate these areas from the Boulder batholith of Weed has helped to perpetuate the impression of the isolation and well-defined limits of that mass.[6]

More recent field work by the Geological Survey[7] has, however, demonstrated the intrusive character and probable Tertiary age of large granite areas both west and east of the original batholith, and while many of the largest outlying masses have not been described, their general relations and character are sufficiently well known to include

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[4] F. V. Hayden: *U. S. Geological Survey of the Territories* (Washington, 1867, 1868, 1869, 1872, 1873).

[5] J. Ross Browne: *Mineral Resources of the United States* (Washington, 1866, 1867, 1868).

Also, R. W. Raymond: *Mineral Resources West of the Rocky Mountains* (Washington, 1870 to 1875, inclusive).

[6] In the past year Prof. A. C. Lawson, of Berkeley, has advanced the hypothesis that the Boulder batholith is in fact a laccolith, occupying a great synclinal basin, and upon this hypothesis has based theories of meteoric origin for its ore deposits. Professor Lawson's paper is based largely upon a review of the literature of the region, and many of his essential premises are not confirmed by field evidence. It can, for example, be definitely shown that the "synclinal basin" is a portion of Rocky Mountain structure several geologic ages earlier than the granite, and is in no sense due to the intrusion of the latter; that the granite has in no place penetrated between the andesite capping and the underlying sediments, but has truncated the sediments below, and only at high points has reached the andesite; and that magmatic chalcopyrite, rather than being indicative of the basic border of the intrusive mass, is closely associated with the siliceous injections of the later segregations. It is difficult to accept Professor Lawson's paper as a candid analysis of the problem.

[7] R. W. Stone: Geologic Relation of Ore Deposits in the Elkhorn Mountains, Montana, *Bulletin No.* 470, *U. S. Geological Survey*, pp. 75 to 98 (1911).

W. H. Weed: Geology and Ore Deposits of the Butte District, Montana, *Professional Paper No.* 74, *U. S. Geological Survey* (1912).

A. Knopf: Ore Deposits of the Helena Mining Region, Montana, *Bulletin No.* 527, *U. S. Geological Survey* (1913).

J. B. Umpleby: Geology and Ore Deposits of Lemhi County, Idaho, *Bulletin No.* 528, *U. S. Geological Survey* (1913).

W. H. Emmons and F. C. Calkins: Geology and Ore Deposits of the Philipsburg Quadrangle, Montana, *Professional Paper No.* 78, *U. S. Geological Survey* (1913).

A. N. Winchell: Mining Districts of the Dillon Quadrangle, Montana, and Adjacent Areas, *Bulletin No.* 574, *U. S. Geological Survey* (1914).

them also in the group. Growing knowledge of the structure and nature of the rocks filling the gaps which appear on the map between the several granite areas, leads also to the belief that these tracts, isolated on the surface, are in reality but the many exposures of a single mass.

The original Boulder batholith remains the largest connected outcrop of this mass, which may be tentatively called the Montana batholith; but large exposures are now recognized to the south and west, while numerous smaller areas indicate its presence at no great depth for a considerable distance to the north and east of the old limits. Gauged by these standards, the granite batholith of Montana has a width of 75 miles, from Philipsburg to Elkhorn, and a length, from Marysville to Dillon, of well over 100 miles.

Batholith in Mountainous Section of Montana

This area is entirely within the mountainous portion of the State, falling between the Rocky Mountain main range on the east and the parallel Bitterroot system on the west. The granite is thus intimately associated in occurrence with the great mountain folds and faults of the Montana cordilleras. These may be structurally divided into several belts, paralleling the northwest course of the ranges.

Most easterly is an outlying zone of gentle folds, which find topographic expression in the Little Belt and Big Snowy mountains. Next on the west is the Rocky Mountain belt proper, carried from the Canadian line to the Yellowstone by the Livingstone, Main, and Big Belt ranges—a belt of closely folded and faulted rocks overriding the eastern zone along great thrust faults. The western border of this zone is indefinite; the folds decrease in intensity, and the mountains drop off into the low levels of the great valleys. These valleys, the Missouri-Jefferson on the east, and Blackfoot-Deer Lodge on the west, beginning on the flanks of the Rocky Mountain belt, and converging southward, surround a rugged plateau, whose area coincides in general with the extent of the main mass of the Montana batholith. To the west, another zone of thrust faulting inaugurates a third mountain belt that includes the complex ranges beyond the Deer Lodge and Melrose valleys (Fig. 1).

The several belts into which the Montana cordilleras may thus be topographically separated coincide with zones of geologic structure. Each belt is in a large way a complete earth fold, anticlinal to the east, synclinal to the west, with Algonkian rocks exposed by the erosion of the former division, and Cretaceous remaining in the basins of the latter. Thrust faulting has caused the anticlines to override the adjoining eastern synclines, and these intensely folded and faulted zones serve to separate the mountain belts.

The time relation of the igneous rocks of the Montana batholith to

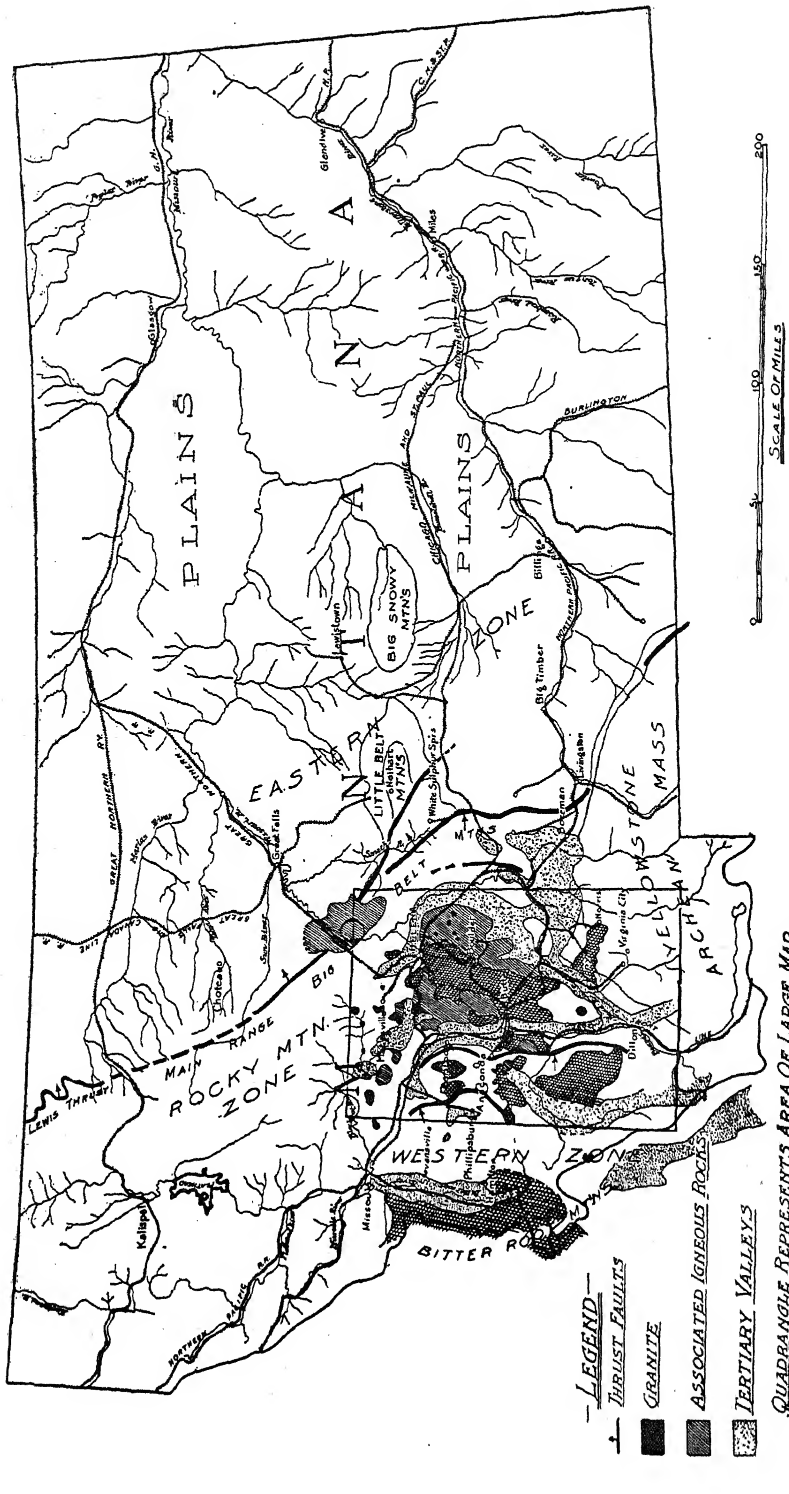

FIG. 1.—SKETCH MAP OF THE STATE OF MONTANA.

this general mountain structure can be fixed with considerable certainty, and is summarized in the following table:

1. Middle Cretaceous.—Main Rocky Mountain folding, and formation[8] of the large earth folds in northwesterly direction.
2. Middle-Upper Cretaceous.—Extensive erosion and beveling of folds. Deposition of terrestrial and shore deposits to east.
3. Upper Cretaceous.—Andesite eruption. Deposition of extrusive lavas and breccias upon eroded surface to west, and formation of tuffs and andesitic sediments to east.
4. Upper Cretaceous(?).—Local intense erosion and formation of coarse andesite conglomerates.[9]
5. Upper Cretaceous(?).—Thrust faulting along northwest lines, and local intensification of folding. Andesitic sediments are included in this movement.[10]
6. Eocene(?).—Intrusion of Montana granite.
7. Eocene.—Extensive erosion. Development of peneplain and well-adjusted river system. Early rhyolite.
8. Oligocene.—Normal north-south faulting, and local accumulation of gravels Erosion of peneplain.
9. Miocene.—Further accumulation of river deposits. Later rhyolite and dacite. Same conditions probably extended through the Pliocene period.

The geological expression of this series of events can be seen in a few critical sections. The first (Fig. 2), along the general line of the Northern Pacific railroad between Helena and Garrison, shows the unconformity between the early folding and the andesite, and the antecedence of the structure to the granite intrusion. The second (Fig. 3), about 30 miles to the south and east, in addition places the thrust faulting after the andesites and before the Tertiary gravels. The coarse andesite conglomerate at Maudlow, near the eastern end of the section, could have been derived from no other source than the lavas of the Elkhorn Mountains. It is, therefore, of later age than the pre-andesite folding, but with the conforming Cretaceous sandstones it has been folded and overturned by the action of the thrust zone to the west. The Lost Creek sections (Figs. 4 and 5), a few miles north of Anaconda, prove the granite to be subsequent to the thrust faulting, while the two remaining sections (Figs. 6 and 7) show the relation of the two rhyolite series to the granite, the gravels, and the erosion surfaces. These are but typical examples. Instances might be multiplied, but the concordance of all evidence found and the absence of conflicting facts leave but little doubt of the essential accuracy of the above tabulation.

The intrusion of the granite was thus a relatively late event in the development of the Montana cordilleras, and the main features of structure had already been developed. The present topography, on the

[8] *Bulletin No.* 470, *U. S. Geological Survey*, p. 78 (1911).

[9] Type locality, hill top southeast of Garrison, where coarse cemented andesite conglomerate lies unconformably upon Kootenai Cretaceous sandstones.

[10] Type locality, Maudlow, Broadwater County.

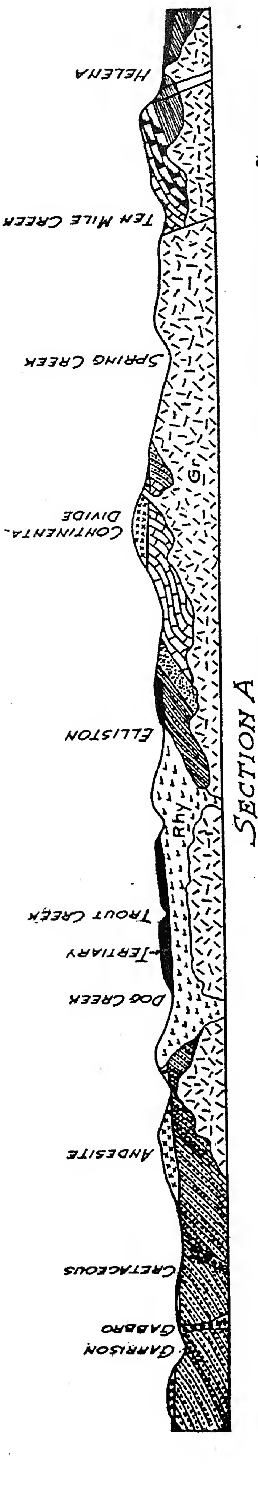

Fig. 2.—Geological Section along the Line of the Northern Pacific Railroad between Helena and Garrison.

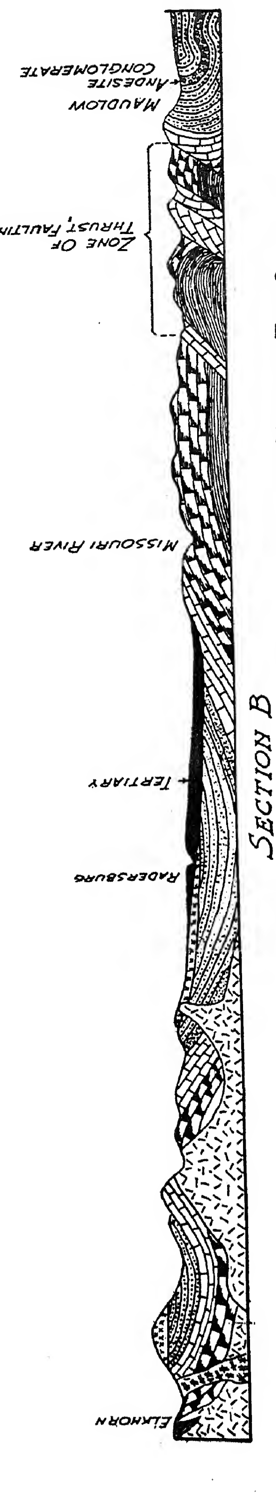

Fig. 3.—Geological Section of Country 30 Miles South and East of Fig. 2.

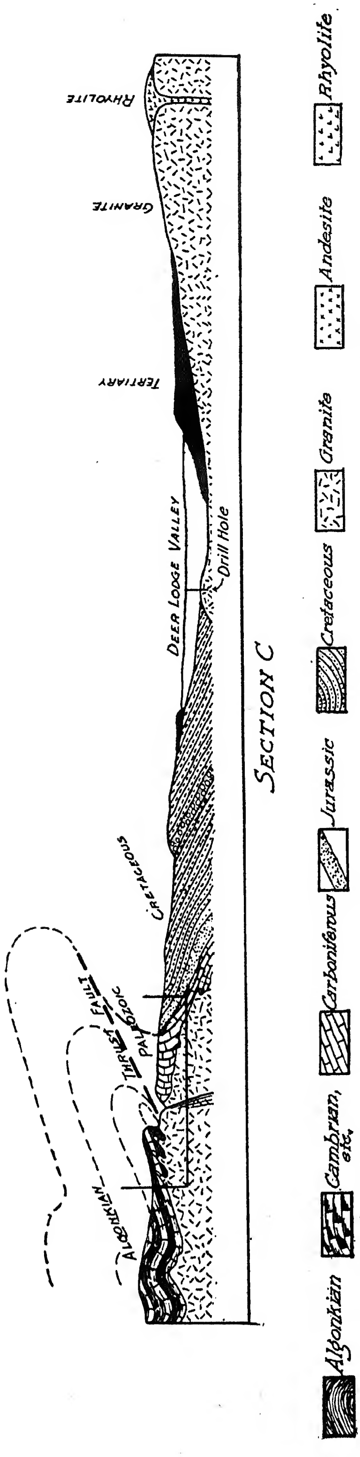

FIG. 4.—GEOLOGICAL SECTION OF THE LOST CREEK REGION, NORTH OF ANACONDA.

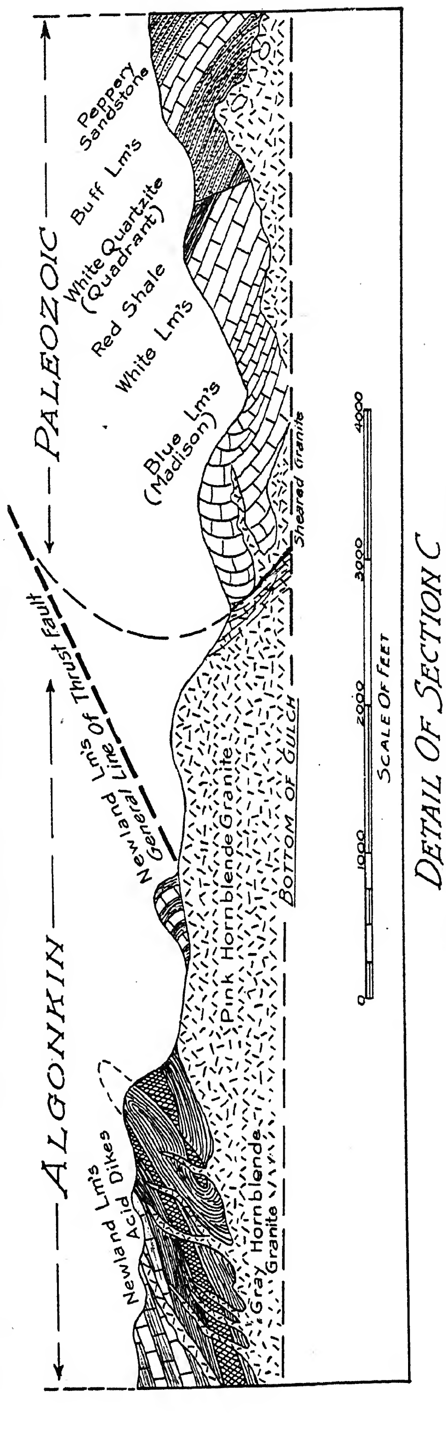

FIG. 5.—DETAIL OF SECTION SHOWN IN FIG. 4.

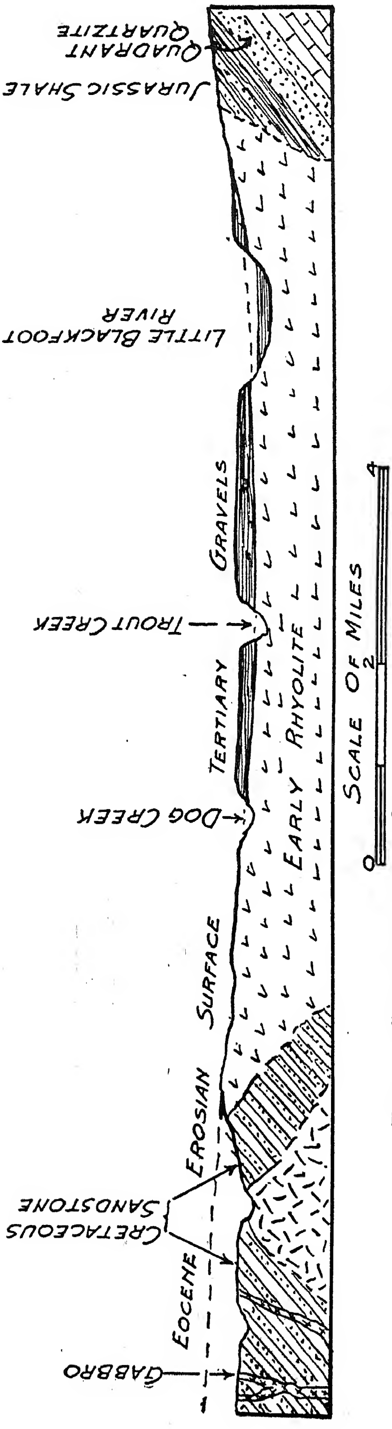

Fig. 6.—Section showing Occurrence of Early Rhyolite, near Avon.

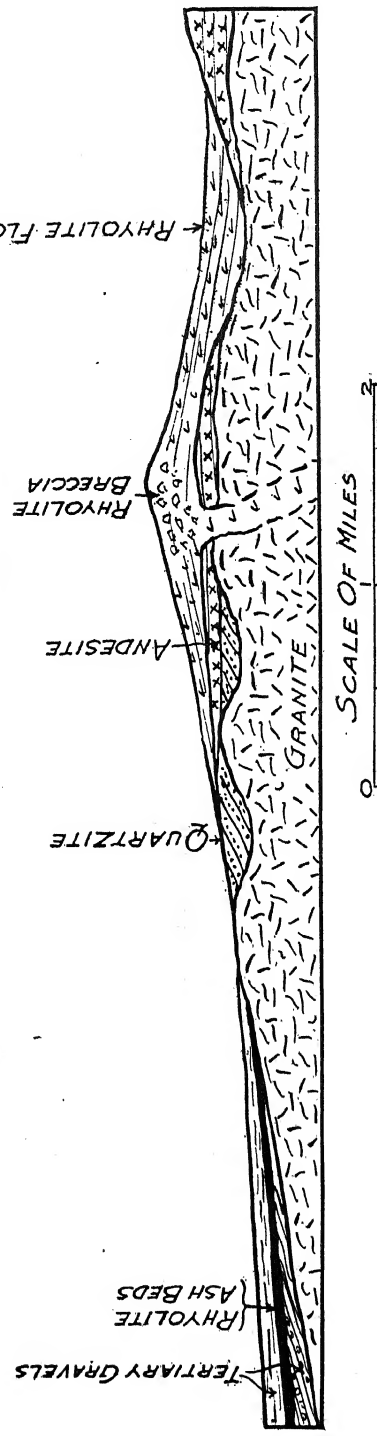

Fig. 7.—Typical Section, East Side of Deer Lodge Valley.

other hand, is of post-granite origin, and the presence of the batholith has determined in no small degree the great valley lines of the mountain region.

Granite Intruded as Dome-Shaped Mass

From fragments remaining of the original cover, and from the position of contacts on the edges of the several granite areas, it is possible to fix within general limits the form of the original top of the intrusive mass. This may be roughly described as a double dome, with high points east of Butte and north of Basin separated by a deep trough. From these crests the granite surface, irregular in detail, slopes gently to the east, north, and west, and is exposed at remote points wherever erosion has

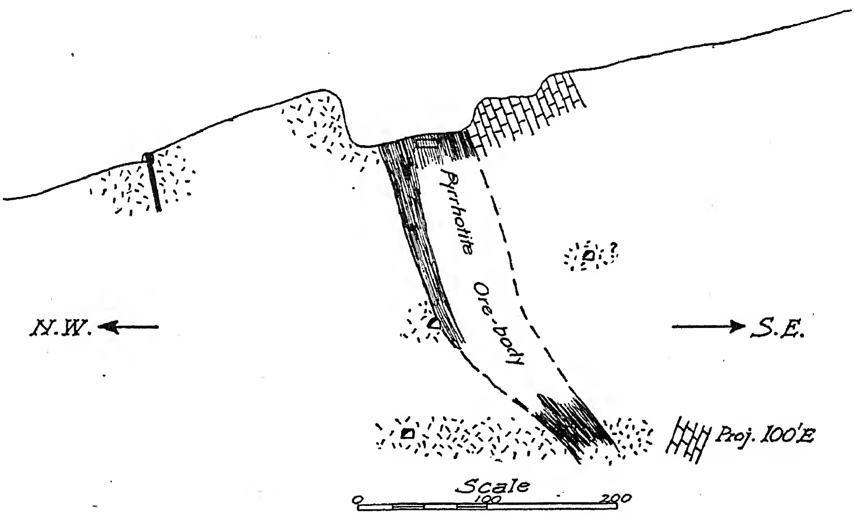

Fig. 8.—Golden Curry, Jefferson County. After Reno H. Sales.

reached low elevations. It is evident that with increased denudation more and more of the granite is being exposed, and its frequent appearance, with normal texture and composition, at low points in the peripheral gulches points unmistakably to its general presence at no great depth. Granite cappings on outlying hills are entirely absent, and there is no evidence of granite overlying the earlier rocks except in the form of local dikes or sills. At many points on the periphery of the main granite area the contact is exposed in deep gulches or in mine workings. Later faulting at many points confuses the contact, but the unbroken downward extent of the granite and the general widening of its borders in depth are universally conspicuous (Figs. 8 to 13).

The absence of any doming or tilting of the sediments by the igneous rock is notable in these peripheral sections. Large areas of folded and faulted rocks are gone, and their place is occupied by granite, but the

change has left no record of dynamic disturbance of the antecedent structure. The removal by erosion of the larger part of the original cover of the batholith has lessened the evidence of the manner of this intrusion, but the general trend of that remaining is clear.

The top of the granite—at the high points of the dome—reached the andesite series, which at that time rested upon the beveled edges of the folded sediments. At lower points a varying thickness of the tilted rocks intervened, and in places, notably northwest of Boulder, west of Butte, and southwest of Elliston, fragments of sedimentary rock remain between the granite and the andesite. These beds retain the structure of the early folds, and the sloping beds are truncated by the granite below. Passing toward the flanks of the dome, an increasing thickness of sedimentary rock appears between the granite and andesite, the

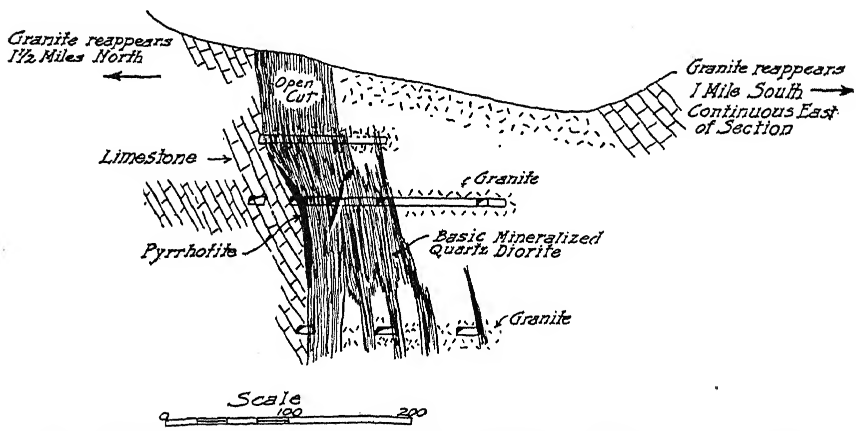

FIG. 9.—SPRING HILL, LEWIS AND CLARK COUNTY. AFTER F. A. LINFORTH.

former showing as irregular intrusions cutting across the bedding independently of structure. It is thus impossible to regard the granite as thrust, in laccolith form, between the andesite and sediments now covered, and it becomes necessary to explain the disappearance of large masses of rock from the area now occupied by the batholith.

The contact effects of the intrusion upon the sedimentary rocks are surprisingly slight. Actual contact phenomena seldom extend more than 1,000 ft. from the igneous rock, and the less susceptible rocks are found unaltered within a very short radius. Recrystallization of the sediments is the most widespread effect of contact action; the addition of iron and silica by magmatic solutions is common; and contact ore deposits of gold-bearing pyrite and chalcopyrite are abundant. In general, pure limestones are recrystallized, and if silica be contributed, tremolite is formed. Magnesian limestones form sandy dolomitic marbles, with development of diopside and chlorite. Siliceous lime-

stones are altered to tremolitic marbles, with contact minerals varying with the impurities of the original rock. Shales form hornstones with or without contact minerals. In quartzites, micas and feldspars are developed, secondary quartz is added, and the original bedding is obscured or obliterated. Andesites show a recrystallization of the ground mass. In all the intruded rocks veins of quartz and specks of pyrite mark contributions from the igneous magma.

The possibility of the fusion or absorption of the sediments into the granite is eliminated by these slight contact effects, as well as by the absence of chemical evidence of such accretions near its borders. The clean-cut truncation of the strata implies mechanical action, the manner of which is suggested by the accompanying sketch (Fig. 14).

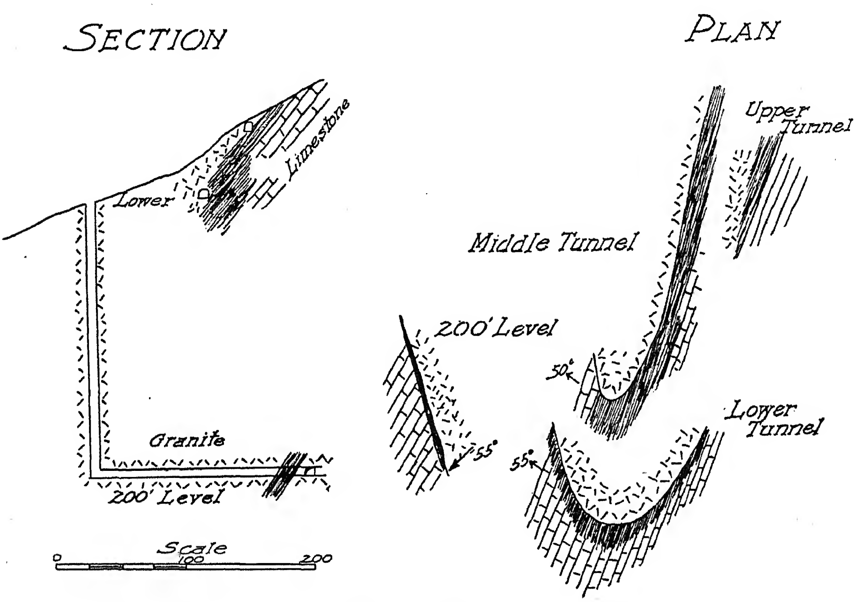

Fig. 10.—Copper Queen, Madison County. After Reno H. Sales.

It appears from this that dikes and sills, vanguards of the intrusion, penetrated into joints and cracks of the sedimentary rocks and broke them into detached blocks. Once loosened, these were pushed aside by the further intrusion of the magma, and finally were either caught as inclusions in the solidification of the granite or subsided through the semi-fluid mass to depths below the present exposed surface. Near the contacts inclusions are common for a few score feet into the granite. These are angular, uncorroded, and often of forms that could be pieced together into their original integral mass. Beyond this zone, however, inclusions are rare, and it seems probable that in the slowly cooling magma a majority of the sedimentary blocks were either absorbed or had

ample time to settle far below the depth so far reached by erosion. This explanation conforms with the theory of magmatic stoping which was applied at Marysville by Barrell in 1907. Field observations throughout the area of the Montana batholith have corroborated its soundness.

Evidence of Increasing Acidity in Magmatic Reservoir

The granite itself exhibits three types of contact phenomena. On the southern edge of the batholith the contact is marked by a profusion of basic inclusions, normally under a foot in diameter, but ranging up to 50 ft. These inclusions are in general fine-grained diorite, slightly more basic than the batholith itself. They show a slight alteration on the

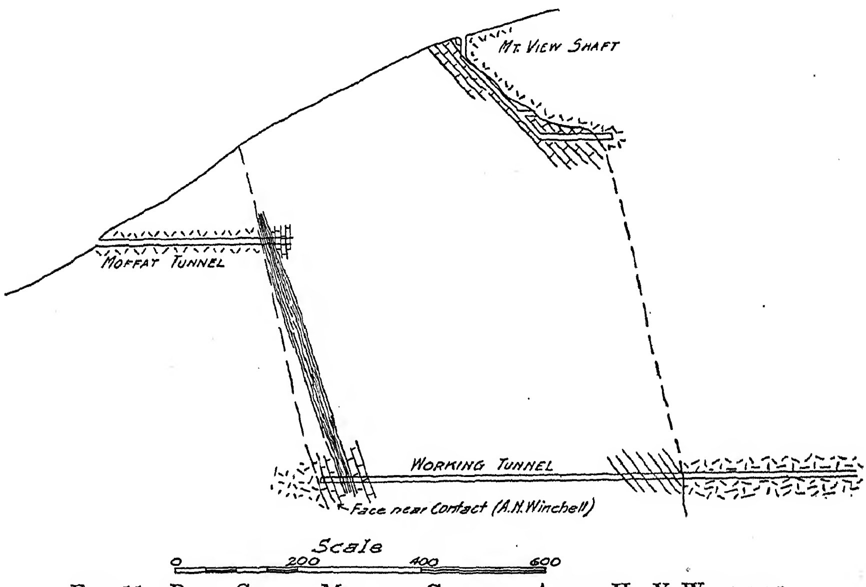

Fig. 11.—Bear Gulch, Madison County. After H. V. Winchell.

edges, are generally of angular form, and are distributed within the granite impartially on limestone, quartzite, and andesite contacts. These facts, together with the absence of any phenomena suggestive of magmatic segregation, lead to the conclusion that the diorite fragments represent an early basic crust of the batholith, caught up and included in a further advance of the main magmatic mass.[11]

On the north, however, the original basic border remains, and in the pyrrhotite and pyroxenite of the Spring Hill mine[12] is found at its extreme development. In this case the distance from the pyrrhotite contact through the gradations to normal granite is about 60 ft., in which

[11] Type locality, near lime kiln, Highland Mountains, Silver Bow County.

[12] Report by F. A. Linforth, Anaconda Copper Mining Co., Geological Department, Butte, Mont. (1912).

interval pyroxenite, gabbro, and diorite successively occur. Similar contacts, less perfectly developed, are found throughout the Helena region, and on granite-andesite borders it is frequently impossible to detect the exact point at which the fine-grained basic granite ends and the recrystallized andesite begins.

The third type is of less common occurrence, and is found only where the more acid phases of the granite form the contact. It takes the form of increasing acidity in the outlying dikes and sills of the igneous rock. These, with crystalline continuity, grade from normal granite at the point of departure from the parent mass to alaskite and even pure quartz at their further terminations.

It seems possible to make these contact types conform to the conception of a general increasing acidity of the main magma. The primary contact was a basic segregation, which in a portion of the periphery still

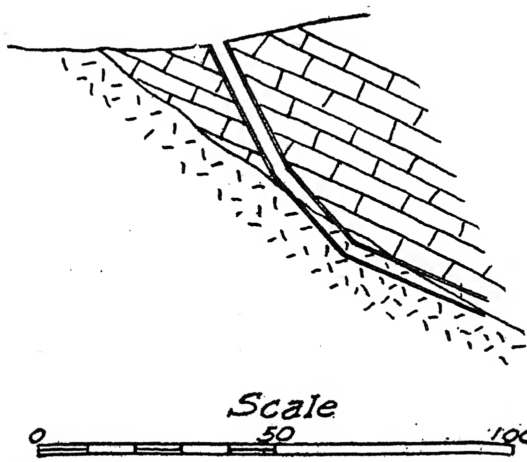

FIG. 12.—IRON MOUNTAIN, ARGENTA, BEAVERHEAD COUNTY. AFTER F. T. GREENE.

exists. Further intrusion on the south pushed the more acid residual magma through and beyond this border, while the more remote penetrations of the igneous rock became increasingly acidic.

The same sequence is evident within the mass of the batholith. The earliest crystallization—apart from the contact—took the form of quartz monzonite, with about 63 per cent. silica. The composition of this varies but little within the limits of the batholith. While still warm this monzonite was intruded by irregular masses of aplite, which, in varying amounts, is of universal distribution and forms roughly, 10 per cent. of the present surface. This contains 75 per cent. silica and bears witness to the increased acidity of the reservoirs. The third stage is represented by siliceous injections within the aplite itself—quartz and jasper veins and stringers that, with the later addition of sulphide minerals, form the typical ore deposits of the batholith. Pyrite

and chalcopyrite are associated primarily with these siliceous veins, but are not rare as primary minerals in the aplite, and are occasionally found in the monzonite itself. Both are of rare occurrence in the basic contact phases of the batholith, and are in general associated with the siliceous emanations of the later periods.

Ore Deposits of Common Origin

The ore deposits within the granite point to some final conclusions. They are typically fissure veins, enlarged by replacement, generally associated with the aplite, and of nearly uniform east-west strike. Many can be traced for several miles, and widths of 50 or even 100 ft. are not

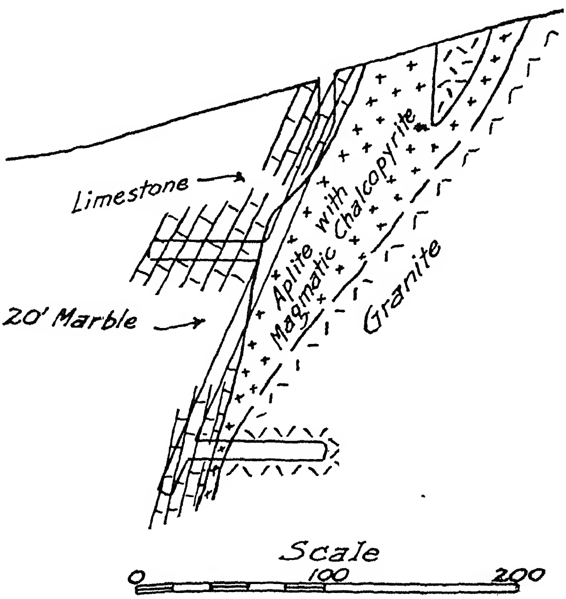

FIG. 13.—MODOC, GRANITE COUNTY. AFTER P. BILLINGSLEY.

uncommon. The widespread and uniform distribution of such veins throughout the granite area bespeaks a common origin, determined by developments within the cooling batholith, and the general association with aplite suggests the ultimate acid residue of the magma as the source of the mineralized solutions.

It is possible to separate the mineral filling of these veins into three periods. The earliest is marked by minerals that indicate the pneumatolytic and semi-igneous conditions then prevailing. Feldspar, vitreous quartz, tourmaline, mica, pyrite, and molybdenite are characteristic. The second period is that of the great sulphide mineralization, with pyrite, chalcopyrite, sphalerite, galena, tetrahedrite, and small quantities of bornite prevailing. This mineralization has afforded the

deposits of the Helena, Wickes, Basin, and (in part) Butte districts. These sulphides are ranged in vertical zones within the veins; galena, with secondary silver enrichment, predominating at the surface; sphalerite most abundant between 400 and 700 ft. in depth; and below only pyrite and sparse chalcopyrite with quartz. This distribution, which conforms to the solubility of the several minerals in cooling solutions, is a universal result of the primary mineralization of the second period. In general all the deposits of this period show all the above sulphide minerals, but it is suggestive of possible variation of the magmatic reservoirs that in the mines north and west of the general line, Butte-Wickes, lead and

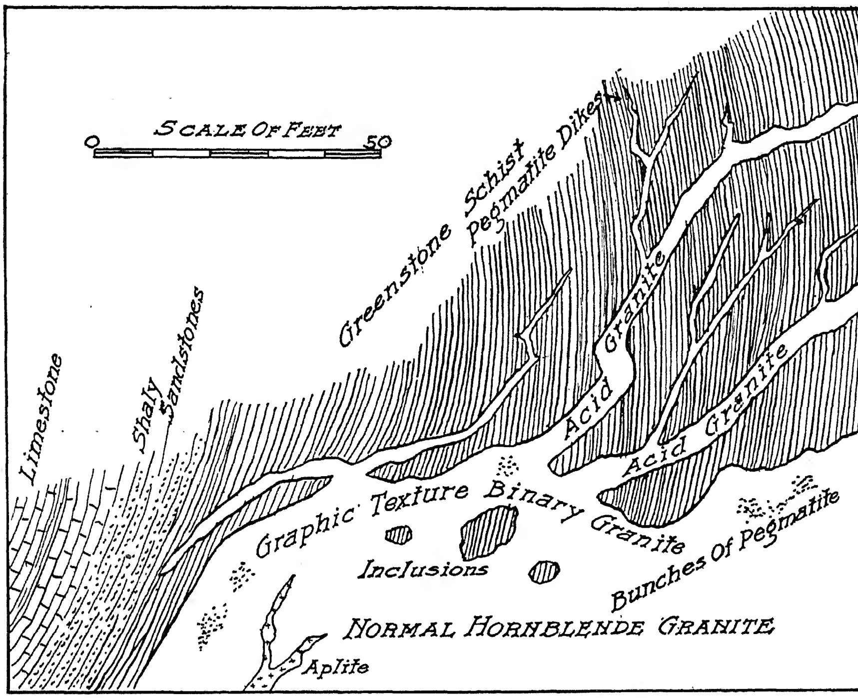

FIG. 14.—DETAIL OF GRANITE CONTACT, LOST CREEK.[13]

zinc, with the secondary silver, predominate, while to the south and east these minerals are in small amounts, with ore shoots of chalcopyrite capped by secondary chalcocite. These orebodies are earlier than the Pliocene dacite-rhyolite, though the numerous dikes may have caused a reconcentration of ore along their flanks.[14]

[13] Camsell (Geology and Ore Deposits, Hedley Mining District) *G. S. C. Memoir No.* 2, 1910, on p. 101, gives the details of a granite contact that affords an interesting parallel to the one shown in Fig. 14.

[14] The prevalence of ore shoots in that portion of the veins adjoining rhyolite dikes suggests this. The Baltimore mine near Boulder, the Comet mine near Basin, and the Hibernia and Nettie mines in Butte are widely separated examples.

The third period of mineralization within the batholith is found only in the Butte district, where a rich copper series—enargite, bornite (rarely chalcopyrite), covellite, and chalcocite—has penetrated from below, and over a limited area replaced the ores of the earlier stage. These border the district, and, with lead and zinc predominating to the northwest and chalcopyrite to the southeast, show the normal vertical distribution of silver, zinc, and pyrite zones. The replacement by copper minerals has proceeded from a center under Anaconda Hill, and has at that point reached the present surface. To the east, north, and west the copper mineralization is found at successively deeper points in the more remote veins, with an increasing cover of earlier ores. This third stage was long continued, the solutions depositing the same sequence of minerals in fissures opened at successive periods. Dikes of quartz-porphyry, found only in the Butte district, may bear a genetic relation to this latest stage, but accumulating evidence of the close association of these dikes with aplite masses in depth leaves this point problematical. The copper enrichment may be placed in point of time between the aplite intrusions in the early Eocene and the rhyolite eruptions of the Pliocene.

The evidence of the ore deposits may therefore be summed up as follows: The intrusion of the aplite and general cooling of the batholith was followed by the development of east and west fissure zones rather uniformly distributed. In the aplite areas pneumatolytic vapors were injected into these openings, with great amounts of silica. Mineralized solutions, differing slightly in portions of the batholith, but representing in general the metallic composition of the magmatic reservoirs, occupied and enlarged the fissure zones by replacement of the walls and deposited the sulphide minerals in a common vertical order. In a single instance a further contribution, in the form of rich copper sulphides, was added. In the absence of any known special factors, the localization of this primary enrichment must be attributed to a segregation of copper within the original magma, and its successive concentration by the splitting off of the several intrusions and solutions.

Conclusions

Some general conclusions regarding the Montana batholith may therefore be briefly enumerated:

1. It is of more widespread extent than the original Boulder batholith, reaching westward with insignificant gaps to Wisdom and Philipsburg.
2. It appears in several of the zones of Rocky Mountain structure.
3. Its age is latest Cretaceous or earliest Eocene.
4. It is subsequent to the folding of the sedimentary rocks, and the dips of the beds on its borders are of no significance.

5. It was intruded as a dome-shaped mass, into folded sedimentary rocks capped by andesite. The high points of the dome, near Butte and Basin, penetrated to the andesite, but on the flanks a varying thickness of sediments remains between that and the granite.
6. The intrusion produced only slight metamorphic effects, either by heat or by contributed material, on the intruded rocks.
7. Structure shows that the sedimentary rocks formerly occupying the area now held by the granite have been removed in mass by truncation of the beds, and detailed evidence points to magmatic stoping as the agency involved.
8. Contact phenomena and the internal succession of rock types point to increasing acidity in the magmatic reservoirs.
9. The ore deposits are of a common type, and indicate the same general conditions of origin, with a magmatic source containing more lead and zinc to the northwest and more copper to the southeast.
10. The Butte district requires a special concentration of copper solutions in the local magmatic reservoir.
11. There is no evidence of downward limitation of the granite, or of external origin for the solutions which produced its ore deposits.

Discussion

James F. Kemp, New York, N. Y.—Mr. Billingsley has spoken so concisely that we may not realize the many important topics which he has passed in review and upon which he has thrown much illumination. The exact determination of the age of the granite, itself, is one. If we were to look over a geological map containing the region of Butte and 25 years old, we would find this coarsely crystalline mass assigned, as were others of its kind, to the pre-Cambrian, or as we used to say, to the Archean. In the course of time it was found to be later. The *Butte Special Folio*, 1907, described it as post-Carboniferous and possibly post-Laramie. Mr. Weed in *Professional Paper No. 74 of the U. S. Geological Survey* (1912) placed it as probably Niocene because of its relations with the andesite, which was believed to be Eocene. Mr. Billingsley now by a very neat demonstration circumscribes it more definitely and we recognize that it must be latest Cretaceous or earliest Eocene. We see, however, that extensive field observations, ably correlated, were necessary to establish the conclusion.

For some years all familiar with Butte have known that there was a central area the veins of which carried copper ores in a quartz gangue; and that outside this area there were veins carrying silver ores in a gangue of manganese minerals. Mr. Sales's able and detailed paper, read at the Montana meeting, August, 1913, not only corroborated these relations,

but emphasized the transitional character of the intermediate or passage zone. Mr. Billingsley is now able to show from observations at increasing depths, the tendency of the copper ores to appear deep down in the veins once regarded as silver-manganese veins. There is some reason to think that the copper ores run in beneath the silver veins as we pass outwardly from the copper center.

In some of the smaller camps away from Butte, apparently the veins carry galena above, base-metal ores beneath, and lean copper-bearing pyrite in depth. As to the cause of vertical distributions of this character, many of us have pondered seriously in later years. The explanation that complex, heated, uprising solutions first lose their lean copper-bearing pyrite, as heat diminishes and pressure wanes; next their zinc, and lastly their galena, seems better than that secondary enrichment or rearrangement in an originally complex, uniform vein has left the galena above, concentrated the zinc below, and the copper-bearing pyrite lowest of all.

Mr. Billingsley has brought out many interesting points as to the relation of the veins when first filled to the original top of the granite mass. They do not seem to have been greatly decapitated by erosion.

Upon one other phase of the batholith additional discussion in a more elaborate treatment would be welcome. Apparently the paper supports the view that the great granite mass made a way for itself by breaking down and absorbing into itself the overlying rock. The process is technically called stoping, the miner's term having been adopted by the geologist. As many of us know, the view has been especially supported by Prof. R. A. Daly of Harvard. We are forced to assume that the detached blocks sink into the molten mass and are absorbed or "digested." We would be glad to know if Mr. Billingsley and his co-worker, Mr. Grimes, have been able to detect anywhere a change in the chemical composition or mineralogy of the granite, from this absorption of outside matter. Is it anywhere visibly more siliceous from absorbed quartzites or more basic from limestone? No more interesting question is before geologists to-day than this one of the upward stoping of batholiths and additional light would be warmly welcomed.

D. C. Bard, Butte, Mont. (communication to the Secretary*).—To one familiar with the area mapped by Messrs. Billingsley and Grimes the amount of careful work involved in securing their results is apparent. The country as a whole is mountainous and not easy of access, so that many a long tramp on foot was required to accomplish the geologic mapping. The gentlemen deserve much praise for completing so well such a difficult problem, done on their own initiative and at their own expense, during week ends and vacation periods.

* Received Feb. 5, 1915.

The theoretical conclusions of the paper are based on carefully observed field data and should in the main meet with the approval of those familiar with the geology involved.

Vertical Sulphide Zones.—One statement of the author, p. 45, I desire to consider. He says, referring to the mineralization in the widely distributed veins of the Boulder batholith, "These sulphides are ranged in vertical zones within the veins; galena, with secondary silver enrichment, predominating at the surface; sphalerite most abundant between 400 and 700 ft. in depth; and below only pyrite and sparse chalcopyrite with quartz. This distribution, which conforms to the solubility of the several minerals in cooling solutions, is a universal result of the primary mineralization." I believe the above statement will not stand amplification. In the first place, I doubt the facts. I know of several prospects in the area where pyrite is practically the only sulphide mineral found in the veins, and some of these veins are high on the mountain tops. In other cases sphalerite and pyrite occur together from the surface down, galena being practically absent. Where galena does occur, I believe, with the author, that it is more plentiful in the upper parts of the vein, but I doubt if this position is due to differentiation in the primary ascending solutions. It is often true that the galena gives way in depth to sphalerite, but I hesitate to accept as a general rule the statement that the sphalerite in turn gives way to pyrite.

But even if we accept the statement of the author that the usual vertical distribution is galena, then sphalerite, then pyrite below, how can he explain the zonal arrangement by cooling of ascending solutions? It is granted that if galena is found at all it is found on top, but these galena croppings are found in the granite in valleys and on mountain tops through a vertical range of 1,500 ft. at least. It would be strange indeed if the accidents of past erosion should conform to temperature variations in the ascending solutions. Furthermore, we have to bear in mind the varying thickness of cover over the batholith at the time of vein formation and the consequent temperature differences in solutions at any given horizon.

I shall not attempt at this place to elaborate a counter theory to explain such zonal distribution of the sulphides as exists; suffice it to say that I think that it is far from a universal condition, and that where it does occur, secondary processes subsequent to the primary mineralization can explain it.

Segregations of Metals in the Magma.—In two connections in the article the author expresses his belief in local segregation of metals in the magma in excess of its average content. On p. 45 he says "it is suggestive of possible variation of the magmatic reservoirs that in the mines north and west of the general line, Butte-Wicks, lead and zinc, with the secondary silver, predominate, while to the south and east these

minerals are in small amounts, with ore shoots of chalcopyrite capped by secondary chalcocite." Again, in speaking of Butte on p. 46, he says "the localization of this primary enrichment must be attributed to a segregation of copper within the original magma, and its successive concentration by the splitting off of the several intrusions and solutions."

I should like to call attention to the fact that there need not be local concentrations of metals within the magma to produce rich orebodies. All that is necessary is a sufficiency of solvents capable of dissolving the normal metallic content of the magma and depositing it locally in concentrated form. Lead veins here and copper veins there need not mean that this part of the magma carried more lead and that part more copper, but rather that in one place the solutions were better adapted to dissolving lead and in the other place copper.

We often hear it said that there must have been a mighty concentration of copper in the magma beneath Anaconda Hill to yield all the copper mined from there. Now the fact is there is more copper in any cubic mile of the basalt that is found near Spokane or Pocatello than has come from the Butte district. If the same favoring solvents had occurred in the basalts as occurred in the Butte granite the Snake River plains could have had Anaconda Hills all the way from Pocatello to the Dalles. The copper is there, but there were no solutions capable of dissolving it and concentrating it into orebodies.

The condition that differentiated Anaconda Hill from Timbered Butte was not so much more copper in the magma under Anaconda Hill, but more solvents to carry it up and localize it in the veins.

Paul Billingsley.—I am familiar with the points raised by Professor Bard and would like to state in brief just where I think they fail.

I will first take the question of the concentration of metals, as that is possibly more fresh in your minds. It seems to me here that Professor Bard's criticism is in the first place based upon a misunderstanding of the sense in which I have used the word "concentration." By this word I refer to the process by which certain portions of the original magma, left behind by the successive differentiations, become relatively predominant in the remaining reservoirs. Thus the copper, for example, originally as widely disseminated in the quartz-monzonite magma as in his Columbia River basalt, was left out of the crystallizations of the quartz-monzonite, aplite, and quartz-tourmaline periods, and was, in fact, "concentrated" into the siliceous solutions that fed the veins of the latest epoch. In the case of the Columbia River basalt the copper was caught in the basic minerals of the basalt itself. It appears to me that the instance of an acidic granite intrusion, passing through after-epochs of aplite, quartz-tourmaline, and sulphide veins, is sufficiently distinct

from that of a basic lava flow to render any further comparison unnecessary.

Professor Bard's statement that varying minerals in veins of different districts are a product of the varying solubilities of their solutions, to me savors strongly of the doctrine of lateral secretion. However applicable elsewhere, this idea seems peculiarly out of place in the case of ore deposits so closely associated with the late siliceous phases of a great igneous intrusion. In the ores of the Boulder batholith I believe the evidence of a direct igneous source is overwhelming.

In the case of the vertical primary zoning of ores I believe Professor Bard has further misunderstood my position. He has stated that he doubts certain facts, but proceeds to show that these facts are not those indicated in my paper. My contention is that a vertical primary zoning—lead above, zinc next in order, and pyrite-chalcopyrite below—is a universal result of the primary mineralization of the second period. Needless to state, the expression of this zoning in the developed ore deposits is subject to the factors of distance below the batholithic cover, secondary enrichment, and varying mineral content of the solutions. I deemed it inadvisable in the limits of a paper on the broader features of the batholith to submit the details necessary to substantiate the fact of this zoning, and in fact I have an additional contribution on this subject in preparation. The following points may, however, be brought forward at present. (1) There are few mines in the Boulder batholith the veins of which outcrop at a greater depth than 1,500 ft. below the original roof. (2) With the exception of Butte, there has been no development on veins to a greater depth than 2,000 ft. below the roof. (3) The great lead-silver deposits of the batholith are in the roof itself or within 1,000 ft. of it. I refer to the Alta, Gregory, Eva May, Comet, Argenta, Bannack, and Hecla mines. (4) The chief base zinc ores are at points where the cover is more remote, as at Clancy and Butte, or on the lower levels of such mines as the Comet. (5) The few mines in the heart of the batholith, at depths of 1,500 ft. or more below the former cover, show a primary mineralization of pyrite and chalcopyrite, with none of the sphalerite and galena characteristic of the mines of the border districts. The Big Major and Bland-Weaver may be cited. (6) In no instance has secondary action removed sphalerite from the sulphide zone of a vein where it was originally present. (7) Secondary action has universally enriched the silver content for a few hundred feet below the surface, so that the upper levels of lead-rich or zinc-rich deposits have alike produced valuable silver ore.

The figures given in my paper—400 ft. depth for the lead zone, 400 to 700 for the zinc—are attempts to show their relative dimensions rather than their exact position in any or every mine.

Waldemar Lindgren, Boston, Mass.—It is clearly shown, of course, that the veins are closely dependent upon the intrusion of the batholith. I may be allowed to say that I think Mr. Billingsley expressed a little too categorically his ideas regarding the arrangement of metals from top to bottom. I think it is probably true, in a general way, but there are many exceptions to it.

There are two points I would bring out now, of the many points involved in this paper, and the first one is in regard to the relation of the silver veins of Butte to the copper veins. I never could understand it thoroughly, and would like to know Mr. Billingsley's opinion as to what relation they have to each other. As you know, they are very different. One is essentially a banded vein and the other a replacement vein without banded deposition.

The second point refers to a question in which I am deeply interested. I would ask Mr. Billingsley (who evidently leans pretty strongly toward the stoping theory, a theory which evidently has a great many strong points), whether there has or has not, in his opinion, taken place any doming of the country by reason of the pressure of the intruding rock? Personally, I am inclined to believe there has been, and am glad to see Mr. Billingsley acknowledge that in his diagram, which does show a distinct doming or uplifting of the sediments. I also wish to call attention to the fact that both Barrell, at Marysville, and Calkins, at Philipsburg, studied minor intrusions, and both have acknowledged that there is distinct evidence of doming of batholith by the rising inclusive mass.

Paul Billingsley.—In speaking about the relation of the silver veins of Butte to the copper veins, I hope it will be understood that my opinions are not necessarily those of my associates in the work in that field, but are entirely personal.

In the section from north to south across Anaconda Hill you find the main copper veins outcropping at Anaconda Hill, the copper coming to the surface except where it is oxidized. Mines a short distance to the north reach this copper through a zone of primary lead-zinc mineralization which increases in depth with distance from the hill. In the more remote veins there are three zones, rhodonite, quartz, and silver occupying the veins above the zinc, and in some of the northern mines reaching a depth of several hundred feet before the zinc is reached. A section across the camp from west to east on any of the main veins shows the same thing. Successive levels of the mines extend beyond the copper into a mineralization of the lead-zinc type, while outlying shafts show above this the additional rhodonite-silver zone.

The copper zone extends its boundaries with increasing depth, but is apparently unconformable with the silver and zinc zones, which pass below into lean quartz, pyrite, chalcopyrite ore. In general the

rich copper sulphides—enargite, bornite, covellite, and chalcocite—have merely enriched the lean quartz-pyrite of this lower zone, but at numerous places they have penetrated into the zinc ores. In such instances the details are irregular, but the copper distinctly is a later addition to a pre-existing mineralization. Beyond question a certain amount of replacement of zinc by copper has occurred, and some migration of the replaced zinc (an idea originated by Murl Gidel, of Butte) is suggested by the presence of impure sphalerite in many of the recent fissures.

The earlier mineralization—silver, zinc, and chalcopyrite—is similar in type to that of the main veins of the other granite districts, and I am inclined to regard it as of the same origin. On this hypothesis the early Butte ores are in genesis and type no different from those formed over widespread areas at a certain stage in the evolution of the batholith. The primary enrichment in copper, however, is a special local feature, occurring at a later point of time and requiring special factors, which are unknown, to explain its presence.

The question of doming may best be approached by a cross-section of the batholith. The granite appears as a dome, with double crests north and south of Basin separated by a deep trough. Above the dome is andesite, which was deposited as flows upon the surface of tilted sedimentary rocks. This surface, which is older than the granite, makes a satisfactory datum plane for determining possible deformation caused by the intrusion. Fragments of this now remain wherever, in the bits of cover left, we find andesite resting upon sedimentary rocks. Above the high points of the dome this old surface has now an elevation of over 8,000 ft. At Garrison, 40 miles west, its altitude is about 5,000 ft. At Radersburg, 50 miles to the southeast, it is less than 4,500. It therefore appears that over the high points of the granite intrusion a pre-existing surface is now found at elevations 3,000 ft. greater than on the flanks of the igneous mass. This may be a coincidence, but it certainly suggests some uplift of the country. There is, however, almost no evidence of dynamic disturbance of the sedimentary rocks at the actual contacts.

Horace V. Winchell, Minneapolis, Minn.—One of the greatest values of papers of this sort is the bringing together of facts which are probably related to each other, but which, if observed singly and reported upon separately, do not appear to have the bearing which they do actually possess. The conclusions here drawn, based upon such a large number of widely separated observations, are, therefore, the more to be relied upon.

I have wondered in listening to Mr. Billingsley whether it was possible to attribute all of the mineralization of Butte to agencies aris-

ing within the one mass of rock, the Boulder batholith, and have noticed the absence of any reference to the quartz porphyry, with which the copper veins of Butte are characteristically associated. The large veins are closely related, geographically, in their strike and dip and general position with quartz-porphyry dikes. In his paper, I believe there is some reference to a differentiation of these dikes, changes of them in depth, but it seems to one who has studied the camp in former years that there must be some genetic relation between the intrusion of quartz porphyry in the monzonite or the batholith itself, and the ores, and that there may possibly be also some relation between the rhyolite and andesite and the silver mineralization; and it requires a rather extended imagination to conceive of the entire mineralization of Butte in the different kinds of ores, the different kinds of veins, both typically banded quartz-manganese veins and the large and extended replacement copper veins, as all to be assigned to one large period of mineralization.

The idea which I think Mr. Billingsley has of the zonal arrangement perhaps leaves us with the impression that every mine in Butte, especially the large mines upon the borders, may present illustrations. His sketch upon the blackboard during his last remarks gave me the idea that when he said that there were 800 or 900 ft. of quartz-manganese and silver ore on the surface, and 1,500 ft. of zinc beneath, it had been demonstrated that beneath this aggregate of 2,400 ft. copper had been found. That does not represent my understanding of the facts. The large mines in which the zinc has been found have not yet turned into copper mines in the bottom, although some copper ore is found in veins which cross the zinc veins, and may belong to a different period.

Paul. Billingsley.—The last point brought up by Mr. Winchell may, perhaps, be answered by naming the Badger State and the Elm Orlu as two mines in which the conditions which I have stated do exist. I did not, however, intend to convey the impression of the general presence of copper in depth. I have purposely refrained from going into definite details in the case of the Butte district.

In regard to the influence of rhyolites on ore deposits, it must be remembered that so far as we know the rhyolite is later than any of the mineralization. In the Hornet mine, near Butte, the rhyolite cuts and displaces one of the main veins of the district. The same is true throughout the batholith. In the Comet mine, near Basin, a rhyolite dike displaces the vein about 50 ft.

I have also been rather chary of mentioning the quartz porphyry, because in the lower levels of the Butte mines, where we should find the lower extension of the quartz-porphyry dikes, we find frequently masses of aplite. The point is unsettled whether the quartz porphyry can be separated from the aplite in many of these occurrences.

In one case Mr. Sales called my attention to a crosscut containing quartz porphyry, while another crosscut 25 ft. away contained aplite. In view of this fact, we do not say much about the quartz porphyry until we find out whether or not it is distinct from the aplite. If it can be proved that it is distinct, it will satisfactorily explain to all of us the latest mineralization in Butte.

L. C. Graton, Boston, Mass.—This question of the distribution of the metals in vertical zones is assuredly one of utmost interest. If Professor Bard intends his conclusions to apply to the zoning at Butte, the only camp in the region described of which I may speak from personal observation, I am inclined to take issue with him and agree with Mr. Billingsley that the zonal disposition of the metals is rather a circumstance of original deposition than a result of later rearrangement by superficial agencies. On the other hand, it seems to me that so far as Butte is concerned there may be another explanation, or possibly a modification of the explanation that Mr. Billingsley has given, for this distinctly evident change in the character of metallic content outward from what may be regarded as the center of mineralization.

In his discussion of this subject I believe Mr. Billingsley has several times used the word "replacement." If by replacement he means simply that one metal may be found at one part of a vein or of the district, while another metal predominates in another part of the vein or the district, no exception can be taken; but if he means that zinc, for instance, was originally present at the spot where copper now is, and that the zinc has actually been driven out by the copper—that is, replaced in the strict sense—I do not feel satisfied by those microscopic studies with which I am familiar that such is the chief explanation of this certainly marked and very interesting zonal arrangement of metals which Mr. Billingsley has described and pictured.

Mr. Winchell has pointed out what seemed possibly to be some idealization of this matter of zoning, to which I have no doubt Mr. Billingsley resorted simply to make the situation perfectly clear. Though my understanding is, of course, not at all comparable with Mr. Billingsley's long and intimate knowledge of the camp, I think Mr. Billingsley would probably have us understand that sharp lines separating zinc and copper, such as he has drawn on the board, actually do not exist, but instead there is a gradual transition, with here and there some overlapping, as indicated by the tongues of copper-bearing ground which in his sketch are seen penetrating into the zinc-bearing region.

This particular subject of zoning is one worthy of much study, to determine whether one metal or mineral actually does come along and to an important degree drive another one out, leaving, perhaps, actual physical remains of the first in greater and greater proportion with in-

creasing distance from the central source, or whether, on the other hand, a progressive change in temperature and chemical equilibrium causes a precipitation of different metals or minerals in gradually changing proportions as part of a single process of deposition. Actual replacement of one set of minerals by another is probably better illustrated at Butte than in most districts. For my own part, however, I am not altogether convinced that such replacement has been the controlling influence in the large-scale concentric segregation of the metals illustrated in this camp.

The detailed paper by Reno Sales upon the ore deposits of the Butte district proper, presented a year and a half ago, excellent as it was, I feel gains, is put in a truer perspective, by this paper of Mr. Billingsley's, which deals with the surrounding region and furnishes, so to speak, the setting of the gem. A very high standard of geological work has been set by these gentlemen in their treatment of the Boulder batholith and its principal locus of ore deposition. They are surely to be congratulated upon this splendid showing, and the Anaconda company commended for the encouragement and support it has afforded through them to the practical application of scientific geology.

The Main Mineral Zone of the Santa Eulalia District, Chihuahua*

BY BASIL PRESCOTT, EL PASO, TEX.

(New York Meeting, February, 1915)

Résumé.—The district of Santa Eulalia lies 12 miles to the southeast of the city of Chihuahua, Mexico. The ore deposits occur in a Cretaceous limestone of unknown thickness, overlain by a series of rhyolitic tuffs and flows. The limestone is very gently folded and is cut by several systems of fissures, of which only those with a general north-south strike are pre-mineral and control the deposition. The deposits are of two main types; the earlier silver-bearing iron sulphide type occurs as high-grade tabular bodies resembling veins and as large masses of relatively low-grade ore. The gangue of this type is composed of lime-iron silicates that point toward a close association with an igneous rock mass which has not yet been encountered. In the later silver-lead-zinc type the ore occurs in vertical chimneys and horizontal mantas, principally as oxides, though at a depth of 1,500 ft. some sulphides have been encountered. During oxidation the ore chambers have increased in size owing to the solution of the walls by the sulphuric acid generated in the process, and a substitution has taken place of nearly one-half part by weight of the original constituents of the ore, by newly introduced elements and radicals. Both types are believed to have come from greater depths than those yet reached by mining, and their ultimate source, from a consideration of similar deposits, is believed to be an igneous rock in contact with the limestone at some horizon below. The deposits are believed to have been formed at unusually great depths, as indicated by the proved great vertical range of the silicate minerals coupled with the absence of the igneous rock, and by the practically unchanged character of the silver-lead-zinc ores as far as developed. The volcanic capping is believed to be distinctly later than the economically important ore deposition.

Introduction

THE Santa Eulalia district is situated in the State of Chihuahua, Mexico, and lies about 12 miles southeast of the capital, the city of Chihuahua. (See map, Fig. 1.) The district embraces the Sierra de La

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Santa Eulalia, an irregular eminence more or less circular in form, measuring from 4 to 6 miles in diameter at the elevation of the mines, with its

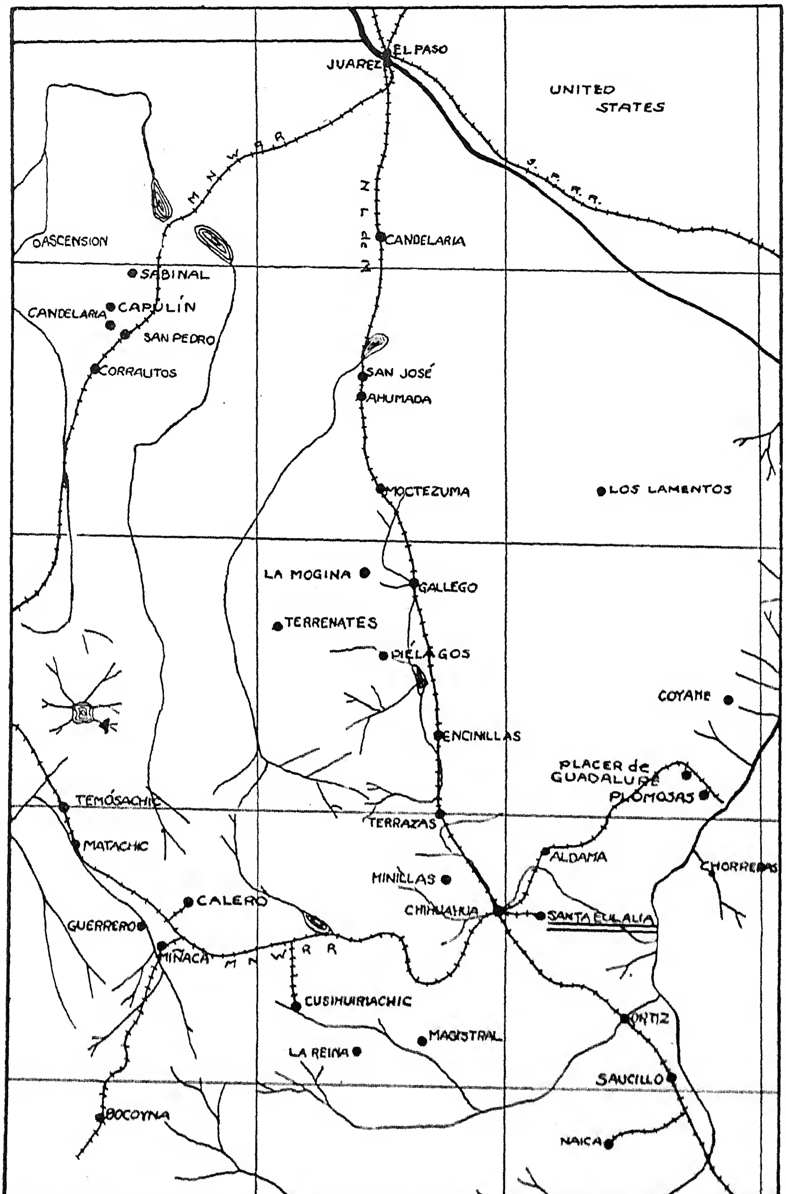

Fig. 1.—Map of a Portion of the State of Chihuahua, showing Location of the Santa Eulalia and Other Mining Camps.

base spreading over a much greater area, and rising to a height of over 7,000 ft. above sea level, or more than 2,000 ft. above the surrounding

plain. The sierra is part of the long, narrow Santa Eulalia range, but is surrounded by such deep, wide passes as to be practically isolated. At an earlier period it was a tableland, or mesa, which still shows in the topography and in the uniformity of elevation of the principal ridges and peaks; but the intense localized erosion typical of the arid climate of the northern plateau region of Mexico has carved wild, precipitous cañons and gorges into the very heart of the mountain. In one of its gentler, less picturesque cañons on the northwestern edge is located the Real de La Santa Eulalia, a typical little Mexican mining village.

The whole mountain is dry and barren, springs are practically unknown, and water is not always plentiful even in the town itself. With few exceptions all vegetation carries thorns or spines, and the only animal life that can survive the inhospitable conditions is composed of the flocks of goats and the scattered burros that in some strange manner exist from year to year.

The Real de La Santa Eulalia is reached from the city of Chihuahua by the Ferrocaril Mineral de Chihuahua, a narrow- (36-in.) gauge road that crosses the plain outside of the city, and, by heavy grades, follows the cañon up to the village. This road has daily passenger service, the trip out taking about 1½ hr., and passes within a short distance of the reduction works of the American Smelting & Refining Co., to which plant it has a spur. The Chihuahua Mining Co. operates a 30-in. gauge road for the shipment of ores, principally those from its mine and those from the affiliated Potosi Mining Co., running from a point near the smelter on the Central Railroad up by the town of Santa Eulalia into the heart of the district, with spurs to several mines. This road is picturesque in the extreme, winding in and out of the cañons, with but little tangent once the town of Santa Eulalia is passed. The Central Railroad of Mexico, part of the National system, has a spur up to a point within a mile or two of Santa Eulalia, and an aerial tramway connects it with one of the mines several miles distant. A second aerial tramway runs out from the Mineral Railroad station at Santa Eulalia into the northern part of the camp.

The history of the Real de La Santa Eulalia and its mines is similar to that of many of the rich silver districts of Mexico, and has been fully and accurately given by many writers. Briefly it is as follows: According to some sources, the deposits were among the first discovered by the early Spanish prospectors, and 1591 is given as the date. However, official records of the State place the discovery at the beginning of the eighteenth century, in the year 1703, or about 12 years after the founding of the city of Chihuahua, a date that does not make the district an old one as Mexican mining districts go. The generally accepted story of its discovery is that it was made by a band of brigands, who, forced out of the mission town of Chihuahua, took refuge in one of the most inaccessible

parts of the Sierra de La Santa Eulalia, and from this point of vantage plied their trade. Such business could not then have been so good as in these last few years, or competition with the Apache Indians was too severe or too dangerous, for when silver and lead were discovered running out of their hearthstones, one night when they had an exceptionally hot fire, they immediately offered the information to the *padre* in Chihuahua in return for complete absolution. The inducement offered the *padre* was that he should be shown silver in such quantities as to allow him to build the greatest cathedral in all America, and while the cathedral of Chihuahua cannot claim that distinction, it certainly is a magnificent example of the architecture and construction of the early Spanish fathers.

The production of the district has been enormous, and though estimates vary widely it probably lies between $300,000,000 and $500,000,000. It seems reasonably certain that the total production for the first 86 years, from 1705 to 1791, amounted to approximately $150,000,000, of which $112,000,000 was reported and taxed.

The height of the bonanza days of the camp was during the last quarter of the eighteenth century, when the Real de La Santa Eulalia counted 6,000 souls, 63 *beneficios*, 168 reduction furnaces and 65 cupelling furnaces, while the city of Chihuahua had a population of 70,000, at least 20 smelting plants, and was practically supported by the activity of its mountain suburb. During the nineteenth century, owing to trouble with the Indians, the withdrawal of the Spaniards, and the political unrest of Mexico, the industry lapsed until the Americans entered the district in about the '80's. A most interesting and enlightening non-technical description of the conditions at Santa Eulalia at the close of the period of French occupation is given by General Lew Wallace in *Harper's Monthly* (November, 1867), while J. P. Kimball,[1] at about the same period describes the geology of the district, the occurrence of the ores, and the costs and methods of mining and reducing them. This paper will always be a classic in the literature of the district.

Following the entry of American mining engineers and American capital, came slowly improved facilities for transporting and reducing the ores; and yet more slowly, improved methods of mining and extracting them. It is a question just how far these improvements have succeeded in reducing costs, which is one of the ultimate objects of all installations of expensive equipment, but they have certainly increased the range of operations, making the deeper horizons as readily accessible as those near the surface; have increased enormously the tonnage that can be handled; and have reduced proportionately the losses during the reduction of the ores. High-grade and low-grade orebodies occur at all horizons, irrespective of the distance from the surface, and the Spanish

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[1] *American Journal of Science,* 2d ser., vol. xlix, pp. 161 to 175, (1870).

and Mexican miners encountered and worked both types. Some of the very oldest mines and deposits are now considered low grade, and, if judged by the ore since encountered in offshoots from the bodies they worked, and by the pillars and walls of ore that they left standing, even giving due allowance for their having extracted the richer portions, the limiting grade in the early days could not have been much above the limiting grade of to-day. This, I believe, is the surprising discovery that has often been made by those who have had occasion to examine old Spanish workings in various parts of Latin America.

Under American management and with American capital, the district has gone ahead until probably at no time in its history has the value in tonnage per year been so great as that of the last few years, and while a few of the mines may have passed their zenith, some of the largest producers look better to-day than ever before, and the district as a whole certainly has not reached its maximum annual production.

Numerous articles on the district have appeared in the technical journals,[2] while the district has been mentioned in connection with other Chihuahuan deposits in many of the papers on northern Mexico. M. A. Knapp[3] gives a description of the eastern portion of the camp, an area not included in this paper; while the results of some of the most complete surveys and studies of the district have never been published. In the present paper, the writer has dealt only with the ore occurrences of the main mineral zone, for, while the rest of the district may be assumed to follow and be governed by the same principles and laws as those that appear to control the deposition within the area described, still it would require careful detailed study to ascertain that such was the case, and there is not sufficient development in the outside areas to warrant such work being undertaken at present.

The main mineral zone as treated in this paper includes the series of orebodies lying approximately N. 10° W., and limited on the south by the southern extension of the Potosi workings, and on the north by the northern limits of Mina Vieja, and including besides these two properties

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[2] Argall, Philip: *Proceedings of the Colorado Scientific Society*, vol. vii, pp. 117 to 126 (1901–04).

Hill, R. T.: *Engineering and Mining Journal*, vol. lxxvi, No. 5, pp. 158 to 160 (Aug. 1, 1903).

Lakes, Arthur: *Mines and Minerals*, vol. xxiii, No. 12, pp. 529 to 531 (July, 1903).

Merrill, F. J. H.: *Mining and Scientific Press* vol. xcviii, No. 1, pp. 37 to 39 (Jan. 2, 1909).

Rice, C. T.: *Engineering and Mining Journal*, vol. lxxxv, No. 25, pp. 1229 to 1233 (June 20, 1908).

Weed, W. H.: *Trans.*, xxxii, 396 to 399 (1901).

[3] *Engineering and Mining Journal*, vol. lxxxi, No. 21, pp. 993 to 994 (May 26, 1906).

the Velardeña, the Santo Domingo, Buena Tierra, and San Toy mines.

General Geology

The oldest formation exposed in the district is a bluish gray limestone. (See Fig. 2.) This is probably Lower Cretaceous, belonging to the great Comanche series that once covered nearly all of Mexico; and while in the United States the formation is not generally of notable thickness, its importance rapidly increases to the southward and it is credited with over 4,000 ft. at the Rio Grande, and with the remarkable thickness of 20,000 ft. in central Mexico.[4] It is the country rock of many mining

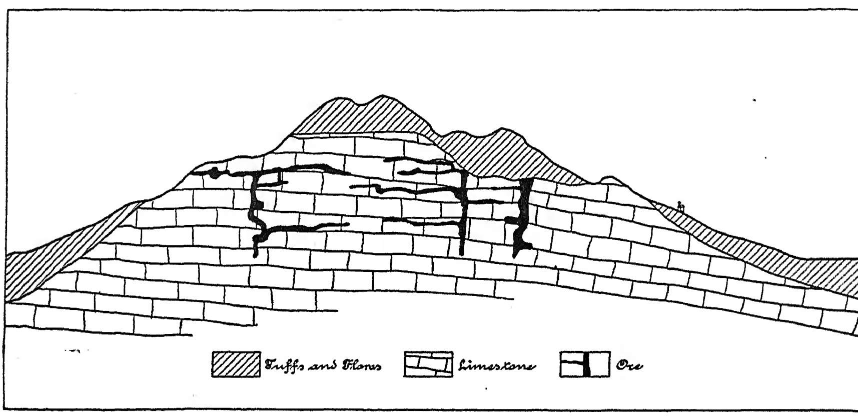

FIG. 2.—GENERALIZED LONGITUDINAL SECTION THROUGH THE ORE ZONE, SANTA EULALIA.

districts in the States of Zacatecas, San Luis Potosi, Coahuila, Nuevo Leon, Durango, Chihuahua and others, and has often a proved thickness as given by combined surface exposures and mine workings of from 2,500 to 3,000 ft., with ultimate depth and amount of erosion unknown. In many localities this limestone is conformably overlain by a shale series of great but unknown total thickness, but certainly often exceeding 3,000 ft., with a transition formation between, marked by several hundred feet of interbedded calcareous shales, and argillaceous limestones. In Santa Eulalia these shales are now lacking, and upon the eroded surface of the limestone is found a deposit of volcanic material and detrital matter, showing a maximum thickness of over 1,200 ft., apparently increasing in thickness as the Sierra Madre is approached toward the south and west, and thinning out near the valley of the Conchas toward the north and east.

The uplift and exposure of the limestone as a land surface took place at the close of the Cretaceous, and probably during the period of great

[4] Hill, R. T.: *American Journal of Science*, 3d ser., vol. xlv, pp. 309 to 323 (1893).

igneous activity that ushered in the Tertiary. In Santa Eulalia this uplift and accompanying igneous activity caused little disturbance of the limestone beds, as dips exceeding 20° from the horizontal are practically unknown, and dips of from 5° to 10° are the rule over by far the greater part of the area. The resulting structure consists essentially of a series of gentle folds with approximate north-south axes pitching very slightly to the south, though at least one monoclinal east-west fold is known. Taken as a whole, moreover, these gentle dips yield a general anticline for the sierra, also with its axis north-south and pitching gently to the south. On the western edge of the sierra the dips are remarkably uniform to the southeast, these continuing back toward the northwest about 1½ miles, at which point they reach the axis and lie practically horizontal or dip slightly to the southwest. This is the most important anticlinal axis of the sierra, as farther to the east the structure consists more of a series of minor folds and these are badly obscured by the capping of volcanic material mentioned above.

In character, the limestone is unusually uniform, there being no marked variation in color, texture, thickness of beds or chemical properties that will serve to distinguish one horizon from another. Fossils of a coarse crystalline calcite have been found at several horizons and, accentuated by differential weathering, have been noted over a large part of the surface of the district. Their distribution through the beds is somewhat irregular, and while variations are not surprising and do not interfere greatly with areal geology, they lead to some difficulties in plotting the conditions found in the relatively small exposures underground. Nevertheless, these fossil beds offer the only known method of correlation of the limestones throughout the camp. Two principal horizons are now recognized. The upper horizon occurs just below the capping in the higher parts of the area and is uniformly rich in fossils at this elevation, but the fossils extend downward irregularly, sometimes to a depth of 250 ft., usually, but not always, being less plentiful in the lower beds than in the uppermost 50 ft. The second horizon has been cut in the shafts and workings of various mines at a depth of about 1,000 ft. below the first, and has not yet been sufficiently opened up to determine its variations. While this horizon normally has a thickness of about 100 ft., it is practically lacking in some places and is nearly 200 ft. thick in others. In many places throughout the limestone, both in the fossiliferous beds and in the intervening non-fossiliferous stratum, chert nodules appear, arranged usually parallel to the bedding, but not sufficiently confined to definite horizons nor sufficiently continuous to be of any value as a factor in correlation.

In general the limestone is compact, semicrystalline, varies in color from light gray to dark bluish gray, and chemically is nearly pure calcium

carbonate. The fossil beds carry sufficient hydrogen sulphide to give a fetid odor.

The uplift of this limestone was followed by a long period of erosion, for not only was every vestige of the overlying shales removed, but even the intermediate series of interbedded shales and limestone are lacking in the Santa Eulalia district. The topographic features at the opening of the Tertiary and the laying down of the first of the volcanics consisted essentially of the same sharp sierras as to-day, but with the flat-topped tableland more prominent and with the cañons radiating from it less precipitous and more sloping than at present. A good idea of the old topography may be drawn from those parts of the district where the capping of tuff has been eroded away and the limestone has been again exposed, this time to the action of the alternate long dry spells and torrential rains. The drainage was essentially then, as now, in all directions from the central portion of the sierra or tableland; but the exact locations of the *arroyos* of to-day are not those of the old surface, nor has the present erosion in the southern and lower areas reached the bottoms of the old cañons, which is perhaps due to the raising of the level of the valley by the addition of vast deposits of detrital matter.

The overlying capping is made up of a series of tuffs and flows, probably from the Sierra Madre to the west, and accumulations of detrital matter from some higher area, probably lying to the north, and since removed by the Conchas river and its tributaries. The series is interesting, but economically and geologically unimportant, and varies greatly in different parts of the sierra. As the district was high land at that time these accumulations were laid down as continental deposits and are not so distinctly stratified as though laid down under water. Nevertheless, they show distinct bedding and occasionally some sorting.

Upon the eroded surface of the limestone, where the topography of that time would permit, an accumulation of a few meters of conglomerate is often found. Above this lies the first tuff, usually rhyolitic, and of a thickness depending upon its location with reference to the old topography, filling first the hollows and cañons between the steep hillsides and tending to make the surface of the land more even, a tendency which, at the same time, was offset by the continual process of erosion. Following this is a flow of rhyolite, again of very variable thickness and again exposing an eroded but more even surface to the next of the series, a mixture of rhyolite and andesite tuff. This deposition of rhyolite and andesite tuffs and flows, interrupted by erosion periods, continued until the total thickness must have been considerable and the old topography was obscured.

At the present time this capping as a whole is not a very resistant rock, certainly not to be compared with the limestone below, and as a result has been rapidly though smoothly eroded. The main flows of rhyolite

and the tuffs, when they have been silicified by later solutions, offer exceptions to the general rule and form topographic features second in ruggedness only to those of the limestone itself.

General Economic Geology

Three distinct types and ages of ore, controlled by fissures, occur in irregular deposits in the limestone. Of these the earliest was a silver-bearing iron sulphide with a mixed lime-iron-alumina silicate gangue, that has been exposed for some years but has only recently become of economic value by the discovery of certain large portions running high in silver. It occurs in enormous masses, much of it too low grade for mining at the present time, but some of it highly profitable. Extending vertically from these ore masses are tabular veinlike feeders, often 100 to 200 m. long by 10 to 15 m. in width.

The second type is the one for which the district is famous. It has been mined for the last two centuries and belongs to the silver-lead-zinc ores common to replacement deposits in limestone. The ore occurs in chimneys and "mantas" (beds or blankets). The chimneys are more or less circular or elliptical in form, often slightly extended along the fissures, though showing all the variations and irregularities common to deposits in limestone. They extend from the greatest depths reached, up to various horizons, and sometimes stand nearly vertical for 1,000 ft. or more. The mantas make off from these chimneys along favorable horizons, lie practically horizontal, and follow roughly the pre-mineral fissures for great distances, often over 1,000 m., to a maximum of perhaps two or three times that distance. The chimneys measure from 20 m. in diameter up to 100 m. in width and 200 m. in length; the mantas from 20 m. or less in width by 10 m. or less in thickness, up to 100 m. in width by 50 m. in thickness, occurring locally even larger, and all variations and combinations being found. With one or two minor exceptions, where square setting was tried by some of the earlier Americans in the district, these bodies have been mined without timber, and as the limestone usually stands well, most of these enormous old stopes are still accessible.

The third type has no economic importance, as apparently workable deposits do not result from it. The ore is a pure galena without admixtures of zinc or iron sulphides, carrying a little silver and practically no gangue, and its deposition took place after the formation of all fissures, and hence long after that of the primary ores of economic importance. It is apparently the result of an expiring phase of mineralization. The occurrence is typically a narrow streak of galena, usually an inch in width, deposited along the fissures of all classes, but best developed in the latest. Local enlargements resulting in small chambers or mantas have not yet been found and are probably entirely lacking.

Both of the economically important types occur as oxides or carbonates, and as sulphides, the sulphide ores greatly predominating in the silver-bearing iron sulphide type; while in the silver-lead-zinc type the oxidized or carbonate ores have been in the past, and are still, the principal producers. Until comparatively recent years the silver-lead-zinc ore was the source of the entire production of the district; but at the present time a small but increasing tonnage of the silver-bearing iron sulphide ores is being produced, perhaps representing one-fifth of the total production of the camp.

Before describing in more detail the two economic types of ore occurrences, their probable genesis and the laws that seem to govern their deposition, a classification of the fissures and dikes and a list of the ore minerals will be given, as these form the basis for many of the statements made.

Fissure Systems

In the limestone, and in certain cases in the overlying igneous capping, are developed certain definite sets of fissures and fault planes. When compared with the fissures found in other mineral districts, these are remarkable for the uniformity of those characteristics which serve to distinguish the fissures of one set from those of another, and, when considered broadly, for their continuity and persistence. This is doubtless due to the uniform character of the limestone, the absence of intense folding, and the position of the fissures—standing nearly vertical or practically normal to the limestone beds. All the various fissures have the characteristic usually shown by those found in limestone of closing and opening with bewildering rapidity, both along dip and strike, though this does not apply equally to all groups.

Classification of Fissures

	Group	Class	Economic Characteristics
Earlier than primary mineralization of economic importance.	1	N–S.	Principal producers of silver-bearing iron sulphide. Large producers of silver, lead, zinc.
		N. 10° E.	Variation of N–S. class.
		N. 10° W.	Principal producers of silver-lead-zinc ore.
		N. 30° W.	Variation of N. 10° W. class.
Later than primary mineralization of economic importance.	2	N. 55° E.	Principal fault planes and channels followed by dikes.
	3	N. 70° W. to E–W.	Principal producers of secondary zinc ore, to date.
	4	N. 30° E.	Principal channels of oxidation.

In the accompanying table the fissures are grouped according to their strike and arranged progressively according to age.

Group 1.—Probably the earliest and the only certain pre-mineral fissures that in no case are later than the ore deposition are those of the North-South group. The direction corresponds with and probably results from the North-South fold, and the fissures are more commonly found along the broad axis of the anticline than elsewhere, though known to occur frequently on either limb. They mark the general direction of nearly all orebodies of the district; no primary ore of economic importance has yet been found that does not occur along one of these fissures. They pass through all chimneys and serve to determine the course of all true mantas over the major part of their distance. None of them has been found passing out of the limestone and into the overlying tuff. They stand nearly vertical, seldom varying 10° from it, even locally, though sometimes, on the limbs of the anticline, there is a tendency for the fissures to stand perpendicular to the beds of limestone rather than vertical. In general the group is remarkable, aside from its productivity, for its tightness and inconspicuousness in the country rock.

This group may be subdivided into four classes. The first or North-South class includes such well-known fissures as the Hematite and Velardeña fissures and those governing the San Toy manta. The second or N. 10° E. class appears usually as a branch or offshoot of the North-South class and is not very continuous either along dip or strike. The third or N. 10° W. class includes such fissures as the Purisima, Los Angeles, Santo Domingo, and Potosi, while the fourth class, or N. 30° W. fissures, is apparently a variation of the N. 10° W. class, caused by a slight irregularity of the anticlinal axis that occurs in the area to which they are confined.

The North-South and N. 10° W. have not been found typically intersecting one another and, from a consideration of the fissures alone, it might seem that they were simultaneously formed, but, as will be shown later, it is probable that the true North-South fissures were earlier. In some cases the different classes occur in the same part of the district, as for example the Hematite and Potosi fissures; but in general each is more or less restricted to predominance over certain areas. For example, very few North-South fissures are found in the northern half of the developed zone, the N. 10° W. fissures being the rule up to the extreme northern limits, where the N. 30° W. class appear almost to the exclusion of other fissures of the North-South group.

It is not an easy matter to determine the continuity of any one of these fissures, both the N. 10° W. and North-South classes affording examples of single fissures that appear to extend over long distances, such as 500 m. or even more, but as a rule the great extent is arrived at by assuming that developed segments will connect continuously. In the upper horizons

many of the fissures represent a series of parallel fractures, no one of which is continuous for more than 50 m., but which as a group and under a single name may be continued for a kilometer or more.

In one or two cases slight fault movements of several meters are known to have taken place along the N. 10° W. fissures after the deposition of the ore, but these cases are relatively rare, and in general none of this group can be said to belong to the type of fault fissures, any noticeable movement being decidedly later than the period at which they were first formed, and representing simply a more or less accidental adjustment of stress along a previously existing plane. The Los Angeles fissure, for example, shows a marked amount of attrition and is accompanied in certain places by heavy gouge and well-developed slickensides. In the central portion of the developed zone, certain N. 10° W. fissures carry a small and variable amount of dike material, apparently representing offshoots from the main dikes, as will be explained later.

The exceptions to the above-noted directions of North-South, N. 10° W., N. 10° E., and N. 30° W. are not numerous; in fact, probably not one case in a hundred will vary from them in the main zone developed by the principal mines. As a rule, when measured over short tangents, the fissures lie within 5° of the given directions, and when the strike is taken over extended distances the variation is even less. It seems probable, however, that in other areas different conditions will prevail, resulting in a slightly changed strike for the whole system, in the same way that the North-West fissures appear to have resulted locally from the N. 10° W. class.

Group 2.—The N. 55° E. fissures form the second group. As a whole this group is characterized by a truly remarkable regularity in strike, for it is nearly impossible to note a variation with a hand compass, and it shows the same uniformity of strike when plotted over long distances. The fissures may vary more in dip than do the North-South group, several of the more important examples dipping from 15° to 20° from the vertical. As a rule they are open for a greater part of their extent than those of the North-South group, and are often filled, especially below an orebody, with products of alteration and secondary deposition. They are the most prominent fault planes of the district, two cases in particular being noted where the movement has probably exceeded 20 ft. and in one case may even reach 50 ft., and innumerable cases are found where the movement has been from a few inches to 10 ft. Most of the dikes follow these fissures, with offshoots from them into the other fissures.

The movements, and apparently the fissures themselves, are later than the primary ore deposition of economic importance, earlier than the oxidation and secondary deposition, earlier than the deposition of the capping, and much earlier than the dikes. Some of the movement, however, was later than the capping and a few cases have been noted where these

fault fissures cut through the lower members of a capping series. The more important faults are accompanied by a coarsely crushed zone several feet wide. Where slightly inclined, the faults, as far as have been observed, are normal, and in the central part of the area are more or less compensating, as the fossiliferous horizons show. This N. 55° E. fault fissuring is exceedingly common, an example occurring every few meters, and while the majority are but knife-blade cracks, well-defined fissures are prevalent.

The exceptions are rare, and it is difficult to determine whether or not they should be treated with this group. A strong N. 45° to 50° E. fissure, called the San Lazaro, cuts the Santo Domingo chimney or *chorro*, which with several parallel associates has some of the characteristics of the pre-mineral fissures. These characteristics, however, are found only in the vicinity of the mineralized North-South group, and it is probable that the fissure itself is not pre-mineral. Although there are no evidences of later movement along these fissures, they appear to cut the capping and possibly should be included under the head of the N. 30° E. fissures to be described later.

Group 3.—The N. 70° W. to East-West fissures form the third group in order of time. These also are fault planes, not usually so marked as the N. 55° E., and probably much more recent. They are not common, although, where found, they are well developed, a few having been traced over distances of several hundred meters and to a depth of 1,000 to 1,500 ft. They are not very constant either in direction or dip, but do not vary far from the vertical. As a rule they represent one of two extremes, either so tight as to attract no attention, the usual case; or so open as to form one of the principal channels for the secondary solutions and for ordinary circulating waters. Although the movements along them have not been determined, they do not appear to have been great, certainly never more than a few meters; but they are accompanied in well-developed cases by a zone of coarsely crushed limestone, which sometimes shows a thickness of from 10 to 15 m. The Gypsum fissure is the best known example of this type. As in the case of the N. 55° E. system, these fissures in rare instances cut the tuffs; but many of the important examples do not do so, and it seems probable that their formation and most of the movement antedate the capping. They are also earlier than the dikes and oxidation and later than the primary mineralization of economic importance and the first two groups of fissures.

Group 4.—The fissures of the N. 30° E. system compose the fourth group. They are later than all other fissures and later than the capping, which they cut, outcropping with remarkable distinctness on the surface of the tuff and porphyry, sometimes for a distance of several hundred meters. They are also later than the dikes, the only dike material found in them being near their intersection with the dike-bearing fissures, and

this representing material, largely clay, washed into them mechanically, as opposed to the occurrence in other fissures where it has been intruded into the open spaces in a molten state. They cut and offset the N. 55° E. fissures, which in turn cut and offset the orebodies of the North-South group. They are fault fissures of very minor movement, seldom showing any crushing of the walls. They are the most pronounced, most numerous and best developed fissures in the district, and in the past, before their barrenness and age had been determined, a great deal of non-productive exploration was done along them. They are of slight economic importance—only as they form channels for oxidizing solutions, and as such may serve as a guide to development. In only one place in the main mineral zone is their relation to other fissures and the ore obscure, and that is in the area before described, where the San Lazaro and associated N. 45° to 50° E. fissures appear in doubtful relation to the orebody. In this area, the trend of the primary orebodies, now oxidized, is approximately N. 30° to 40° E., apparently following one or both of these two sets of fissures out of the Santo Domingo chimney; but irregularities in the mantas in the vicinity of the main chimneys are the usual occurrence. Judging these fissures by the remainder of the district and the manner in which they intersect other groups, it seems probable that the ore deposition was controlled by some other factor than the course of these fissures.

The Dikes

Two classes of intrusives are found in the zone, andesitic and rhyolitic. They do not occur very abundantly nor are they well developed. Unfortunately, also, they have not as yet been found intersecting, hence their relative ages have not been determined. Their relation to the capping does not afford the desired information, as they both cut the capping series and outcrop on the surface. The relation of both to the ore deposition is the same; that is, decidedly later than all primary deposition of economic importance, later than all fissures except those of the N. 30° E. group, and earlier than the oxidation and alteration of the orebody. Both the andesite and rhyolite dikes followed the pre-existing N. 55° E. fissures as main channels, but at their intersection with the North-South group and the N. 70° W. to East-West group they are often found following these fissures for distances up to 100 m. In fact, they show conclusively which fissures, water courses, caves, etc., were open at the time of their intrusion, as they apparently filled every available open space.

The andesite dikes are comparatively fresh and unaltered on the surface and in those parts of the mines far removed from orebodies, but in the vicinity of the oxidized ores are almost completely altered to kaolin, probably by the action of sulphuric acid solutions. They occur both at the northern and southern extremities of the developed area, in the latter,

the Potosi mine, reaching their maximum development in a single dike nearly 20 ft. wide.

The rhyolite dikes occur throughout the developed area and are characteristically smaller, seldom exceeding a meter in width, and are even more intensely altered than the andesite dikes. A fairly fresh sample from the Potosi mine gave an analysis closely approximating that of normal unaltered rhyolite.

Ore and Gangue Minerals

In Table I the ore and gangue minerals are listed in four groups, the first three corresponding to the three types of ore, and the fourth, in limestone, including those minerals developed in the channels of the limestone, often so far removed from their source that it is difficult to refer them definitely to any type. Alunogen, occurring as an efflorescence on limestone walls, is an example, for the sulphate radical may be derived from the oxidation of any type of ore, and the alumina from either the minute quantities contained in the limestone, from the silicate gangue of the silver-bearing iron sulphide type of ore, or from the alteration of the dikes. The subdivisions, primary and secondary, distinguish the minerals of the sulphide ore from those of the oxide or carbonate ore—that is, those formed by ascending solutions from those formed by descending solutions—with the one exception that the quartz, which was introduced after considerable of the oxidation was complete, probably came from below.

The list is certainly not complete, since the purpose of the examination was not such as to warrant the investment of time in the search for and determination of mineral species. However, the list probably contains a sufficient number of critical minerals to classify the deposits according to our present ideas of economic geology.

These minerals are all common and well known, excepting perhaps ilvaite, fayalite, and kenebelite, and it is probable that the field would prove an attractive one to a mineralogist in search of the rarer minerals.

The Silver-Bearing Iron Sulphide Type Ore

There are very few known occurrences of the silver-bearing iron sulphide ore in the Santa Eulalia district, and these have all been encountered within the last 10 or 15 years. Several factors have, unfortunately, retarded the development of the type: First, the ore where first encountered was largely of a grade that at that time offered little inducement to further development; second, the occurrence was considered as a local phase of the silver-lead-zinc type, and as such was of little importance; and third, where encountered in the early stages of development, it was often intermingled with the later type of lead-silver-zinc ore, and its true nature

TABLE I.—*Occurrence of Ore and Gangue Minerals*

	Silver-bearing Iron Sulphide Type		Silver-lead-zinc Type		Pure Galena Type		In the Lime-stone
Pyrrhotite	Prim.		Prim.				
Pyrite	Prim.	Sec.	Prim.	Sec.			
Marcasite							Sec.
Arsenopyrite	Prim.						
Magnetite	Prim.						
Hematite		Sec.		Sec.			
Turgite		Sec.		Sec.			
Limonite				Sec.			
Copiapite							Sec.
Melanterite							Sec.
Siderite			Prim.?	Sec.			
Manganite			Prim.?	Sec.			
Psilomelane			Prim.?	Sec.			
Wad				Sec.			
Pyrolusite				Sec.			Sec.
Rhodocrosite				Sec.			Sec.
Rhodonite				Sec.			Sec.
Sphalerite	Prim.		Prim.	Sec.?			
Smithsonite				Sec.			Sec.
Calamine				Sec.			
Hydrozincite				Sec.			
Goslarite							Sec.
Willemite				Sec.			
Greenockite				Sec.			
Galena	Prim.	Sec.	Prim.	Sec.	Prim.		
Anglesite				Sec.			
Pyromorphite				Sec.			
Mimetite				Sec.			
Cerussite				Sec.			
Wulfenite				Sec.			
Knebelite	Prim.						
Chalcopyrite	Prim.		Prim.?				
Chalcocite				Sec.			
Brochantite				Sec.			
Malachite				Sec.			
Azurite				Sec.			
Native silver				Sec.			
Cerargyrite				Sec.			
Argentite	Prim.		Prim.				
Proustite	Prim.						
Gold	Prim.						
Mirabilite							Sec.
Epsomite							Sec.

TABLE I.—*Occurrence of Ore and Gangue Minerals.—Continued*

	Silver-bearing Iron Sulphide Type		Silver-lead-zinc Type		Pure Galena Type		In the Limestone
Alunogen	......	......	......	......	.....	......	Sec.
Halite	......	......	......	......	......	......	Sec.
Barite	......	......	Prim.	......	......	......	......
Fluorite	Prim.	......	Prim.	Sec.	......	......	......
Calcite	Prim.	Sec.	Prim.	Sec.	......	......	Sec.
Aragonite	......	......	......	......	......	......	Sec.
Dolomite	......	......	......	Sec.	......	......	Sec.
Gypsum	......	Sec.	......	Sec.	......	......	Sec.
Sulphur	......	......	......	Sec.	......	......	......
Quartz	Prim.	Sec.	Prim.	Sec.	......	......	......
Chalcedony	Prim.	......	......	......	......	......	......
Ilvaite	Prim.	......	......	......	......	......	......
Hedenbergite	Prim.	......	......	......	......	......	......
Chlorite	Prim.	Sec.	......	......	......	......	......
Fayalite	Prim.	......	......	......	......	......	......
Uralite	......	Sec.	......	......	......	......	......
Tremolite	......	......	......	.....	.....	......	Sec.

thus obscured. Several years ago, however, the discovery was made that great tonnages of this type carried high silver percentages, making such ore extremely desirable, and recently a clearer conception of the occurrence and characteristics of the type was obtained, with the result that in the future it should be an increasingly important factor in the production of the district.

The following partial analysis of a sample of the ore, made by W. E. Soest for this purpose, gives a good conception of the character of this type, even though the sample itself may not be the average occurrence:

Pb	Zn	SiO_2	CaO	$BaSO_4$	Fe	Mn	S	As	MgO	Al_2O_3	Total
0.2	0.5	11.2	4.6	1.4	39	5.1	14.6	2.5	1.2	3.6	83.9

Recalculating this analysis on the basis of its known mineralogical content, gives: Galena, 0.23; sphalerite, 0.74; arsenopyrite, 5.43; pyrite, 3.75; pyrrhotite, 26.77; barite, 1.4 per cent.; the remainder, consisting of Fe, 19.69; CaO, 4.6; Mn, 5.1; MgO, 1.2; SiO_2, 11.2; Al_2O_3, 3.6 per cent., cannot be readily recalculated.

The conditions in Mexico during the last year have been such that a microscopic examination of this ore has not been made, as planned, but an examination of a number of specimens shows the presence of several rare silicates. An analysis of one of these by W. E. Soest gives: SiO_2,

28.9; Fe, 38.9; CaO, 10; Mn, 4.2; and H_2O, 2 per cent., which checks the previous determination of ilvaite. Under the microscope this mineral is opaque in the thinnest section obtainable, but fine-crushed fragments show the typical pleochroism of ilvaite. Magnetite occurs throughout the sections examined so finely divided that it was not previously recognized in hand specimens, and associated with it is the ferrous manganese silicate knebelite and the ferrous silicate fayalite. As the latter has not been reported in ore deposits, great care was taken in its determination, and a number of thin sections were examined in which fayalite was the most prominent mineral. Its index of refraction, determined in fragments immersed in various liquids, is greater than 1.83 and less than 1.93, and its birefringence, tested with a quartz wedge in sections showing some quartz, is about 0.05. An analysis by William J. Van Sicklen shows SiO_2, 28.3; FeO, 55.6; MnO, 15.3; CaO, 0.0; total, 99.2 per cent. Another analysis, by W. E. Soest, shows SiO_2, 29.1; FeO, 49.1; MnO, 19.1; CaO, 0.1; total, 97.4 per cent. Other tests show even greater variation in the iron and manganese, due, probably, to isomorphic mixtures of knebelite and fayalite. In some cases it has been subjected to hydro-thermal alterations resulting in a brownish mineral somewhat resembling iddingsite, which could not be determined, chlorite, chalcedony, and magnetite. The chlorite, however, usually occurs as a direct replacement of the limestone, and although the magnetite is invariably associated with the fayalite and knebelite, the latter are seldom altered. These minerals are not present as rare constituents but as a group normally form nearly half of the ore, each locally predominating, and each probably aggregating thousands of tons.

From a consideration of the mineralogical content and chemical analysis, the occurrence might well be included under the head of contact-metamorphic ore deposits, presumably with limestone for one wall and an igneous rock for the second; and from the low amounts of alumina, the great quantities of the basic lime-iron silicate, ilvaite, the absence of copper sulphides, and the presence of argentiferous pyrite and arsenopyrite, it might further be classified as belonging to the last stage of contact metamorphism. On the other hand, the presence of the large quantities of magnetite and fayalite would seem to point to the intense physical conditions found only in a magma or in close association with it. Although the specimens examined microscopically came from a depth of about 500 ft. below the contact between the tuff and limestone and at least 600 ft. above the bottom of the mine, no igneous rock has yet been found in the main zone aside from the few narrow dikes unaccompanied by metamorphic action and the volcanic detritus of the capping.[5]

[5] In the eastern camp, there is a stronger dike with silver-lead-zinc ore along its contact but unaccompanied by contact-metamorphic minerals.

In shape, form, and dimensions, the type shows considerable variation, but the economically important occurrence is a tabular, veinlike body, 5 to 15 m. in width, standing vertical and developed downward 500 ft. below the capping, with ultimate limits in depth unknown. The walls are parallel and show as little variation and irregularity over great distances as do the walls of the average vein. The ore is usually frozen to the limestone, but there is no intergrading of the ore with the wall rock and in the economically important occurrences no noted development of silicate minerals in the limestone.

The ore itself is usually banded vertically, parallel to the walls, apparently not developed as incrustations in an open fissure, but as mineralization along parallel fractures of the same fissure with later replacement of the included country rock; and where this replacement has not been entirely completed and the white limestone horses, a foot or more in width, are included in the dark ore, the banded appearance of the orebody is very strongly emphasized.

All the ore of this type found so far occurs along the true North-South class of fissures and there is reason to believe that mineralization of the silver-bearing iron sulphide type is confined to them to the exclusion of the other classes of the North-South group; although definite proof is not yet obtainable, it seems probable that this mineralization usually antedates the formation of most of the other fissures which, when considered from the standpoint of the silver-lead-zinc type, are pre-mineral. The characteristic occurrences of economic importance extend along these North-South fissures for 100 to 150 m. and sometimes much more, and then thin out with remarkable suddenness, an orebody of nearly average width being sometimes reduced to a scarcely perceptible fissure within a couple of meters. Along the same fissure, however, several orebodies may occur, widely separated by stretches of barren limestone in which the fissure itself may be practically imperceptible and indistinguishable from the numerous, small, unmineralized breaks of the limestone.

Bodies of oxidized ores of this type have been found, but these are relatively rare, as the character of the mineralization and the density and texture of the ores are not favorable to rapid oxidation, and the type is considered from the standpoint of sulphide only. The known oxides were originally low in gangue and high in metallic sulphides, so that the resulting ore is predominately an oxide of iron, usually hematite; but a few minor occurrences of the semi-oxidized silicates show that the pyroxene usually breaks down into amphibole, a soft brownish uralite, while the alteration products of the more resistant ilvaite have not yet been isolated.

The principal variations from the above type occurrences are found usually where very favorable limestone horizons are crossed by the

ore channel. Working down on one orebody, it was found to widen rapidly from the normal occurrence until, within something over 100 ft. of additional depth, it was nearly as wide as long, yielding a great tonnage of excellent ore. The most interesting and complex occurrence of this type of mineralization is an enormous mass not yet completely developed, but showing at present 1,000,000 tons or more of mineral matter. Portions of this mass are high grade; the remainder is simply a limestone, replaced by silicates, carrying a small proportion of metallic minerals. The feeders of this orebody are the North-South fissures, several in this case, marked by the vertical banding of their contents, but this banding is found standing at increasingly flat angles as the fissures are left, until the greater part of the area is a horizontally banded, completely metamorphosed mass with the width nearly equal to the length, and a depth, outside of the feeding fissures, much less than either length or width. This mass connects continuously with one of the high-grade tabular bodies and has itself high-grade portions, and furthermore in places has been enriched by the addition of impregnations of the silver-lead-zinc type from a chimney of that ore which touches this body on one edge. This admixture was the ore first encountered, and it is not remarkable that the occurrence should have been considered as merely a local phase of the silver-lead-zinc mineralization, as it appeared to grade into a typical body of that type. A careful examination shows, however, that this silver-lead-zinc mineralization was introduced later and was distinct from the main development of metamorphic minerals and deposition of iron sulphide, for, particularly where the enrichment has not progressed far, it may be noted that it consists largely of a filling of open spaces in the pre-existing ore. The body, since its formation, has been cut by all the post-mineral fissures of the various classes, and these stand vertically and have slightly shattered the ore for a few inches in width, rather than making simple fissures as in the limestone; and these shattered zones have been filled with calcite and secondary quartz, rhodochrosite, pyrite, sphalerite, and galena, perhaps in the order named, this mineralization being secondary and later than the enrichment above described, for these fissures cut through the portions of the orebody enriched by the lead-silver-zinc type as clearly as they do through the unenriched metamorphic ore. These fissures are locally very plentiful and give to the deposit a false impression of vertical banding, similar to that found in the North-South feeders, particularly as some of the more soluble lime-iron silicates have been rearranged parallel to and along the edges of the fissures. Continued examination shows that the silver-bearing pyrite type, locally high in gangue and low in metallic content, lies horizontally bedded at all points excepting along the North-South feeder fissures, and that the cross fissures cut this banding at right angles.

In summation, it is believed that this type of ore is the earliest miner-

alization yet exposed in Santa Eulalia; that a time interval elapsed between its deposition and that of the silver-lead-zinc type; that all cross fissures were later than the mineralization; also that the ore rose along the fissures practically as veins, though not continuously, there being intervening barren sections. On certain favorable horizons the mineralizer made off into the surrounding rock, metamorphosing it into a low-grade body of silicate ore which in some cases has been enriched by additions of the later silver-lead-zinc type.

It is important that this type of ore be distinguished readily from, for instance, the low-lead class (see below) of the later silver-lead-zinc type, for the differences in the line of development pursued are so great that the development of one does not accomplish the development of the other. This distinction may be readily made even on a small exposure, by the presence of arsenopyrite in detectable quantities in the metallic content of the earlier type; the presence of dark silicates or their alteration products in its gangue; by its prevalent vertical banding in the pay ore as opposed to the almost universal horizontal banding of the lead-silver-zinc type, and, if sufficiently developed, by its tabular form.

The Silver-Lead-Zinc Type Sulphide Ores

Clean primary sulphide ores of the silver-lead-zinc type are nearly lacking in the district up to the depths yet reached. Several chimneys show undeveloped sulphide ores in their lowest horizons and one of these occurrences has been developed for several hundred feet, but this body has been partly inaccessible for some time, and has oxidized with surprising rapidity since it was first exposed. There is another occurrence of sulphide ore in the district which has every indication of being primary, and unenriched by secondary solutions carrying metals, but this, unfortunately, had not been completely developed at the time of this examination, so that its general nature, extent, and relation to other nearby bodies of carbonate ore could not be determined. Strangely enough, this occurrence, instead of being in the deepest explored horizons, is found but a short distance below the capping, in places separated from the volcanic material only by a comparatively minor occurrence of oxidized ores. It has been noted, however, that the occurrence of sulphides depends almost entirely upon the local channels of circulation of the oxidizing waters, rather than upon the depth below the surface, and one of the factors governing these waters is the thickness and condition of the more or less impervious capping. It has been further noted repeatedly that near an open channel, where the waters have had free passage, instead of being completely oxidized, the ore may be partly altered, the supposition being that the unretarded waters did not then percolate through the whole mass as they did when the channels were more devious and

less well defined. In this case, it seems probable both that a nearby N. 70° W. fissure offers the open channel that drains the oxidizing solutions traveling along the contact away from the area, and that the unfissured condition of the impervious capping at this point protected the body from above.

Some difficulty is encountered when the reconstruction of an average normal primary sulphide ore is attempted. The commercial analyses of shipments to the smelters and the results of systematic sampling are the only records covering a sufficiently great quantity of ore to give an idea of its average content, but these analyses are run only for commercially important elements and oxides, and the samples often include wall rock, usually include portions somewhat oxidized, and often portions high in secondary sulphides. The occurrences of sulphide ores are not numerous nor in all cases accessible, and their variations are considerable. It seems possible that certain of these variations are due to the depth at which the ore is found, so that even if a clean primary ore were encountered at a deep horizon, it is doubtful if this would represent the original ore of the oxidized bodies found above.

The following are partial analyses of the sulphide ores, No. 1 representing a composite of many samples and the work of many assayers, Nos. 2, 3, and 4, analyses made by W. E. Soest for this work, and No. 5 a selected composite of several samples, analyst unknown.

No.	Pb Per Cent.	Zn Per Cent.	Fe Per Cent.	SiO_2 Per Cent.	Mn Per Cent.	CaO Per Cent.	S Per Cent.	Total
1	15.	14.	28.	4.	0.8	2.6	24.	88.4
2[a]	13.3	12.	22.7	4.	2.1	10.7	20.5	87.6
3	2.	4.	17.	16.	1.5	10.	25.	75.5
4[b]	26.6	12.5	30.	0.4	0.5	0.4	28.2	99.2
5	12.7	10.3	36.	2.2	1.	1.5	28.1	91.8

[a] Also: As, 1.3; Mg, 0.8; $BaSO_4$, 0.2

[b] Also: As, 0.2; Al_2O_3, 0.4.

No. 1 is an average analysis of one of the few large sulphide bodies now exposed in the district, but when it is considered in detail it is immediately evident that the ore is far from being primary and unaltered. It is useless to attempt to recalculate the analysis, as the sulphur is insufficient to satisfy the iron, lead, and zinc, as sulphides, of any known mineral species; but the ore is believed to be composed of galena, sphalerite, and pyrrhotite—the latter to the practical exclusion of pyrite—sulphates of the same metals, a trace of arsenopyrite, and a gangue of quartz and mixed carbonates with a little oxide of iron. This would satisfy the

analysis, but the result is an ore on which oxidizing processes have already commenced. It is unfortunate that this body could not be inspected, for only a few specimens were available. The undetermined elements in this and the other samples usually consist of unimportant amounts of arsenic, magnesium, barium, aluminum, fluorine, chlorine, phosphorus, and antimony.

No. 2 represents an aliquot part of a number of shipments of sulphide from another orebody. It is very similar in mineralogical character and chemical composition to No. 1, except that it shows the results of the admixture of small amounts of wall rock, which lowers proportionately the metallic sulphide and raises the lime.

No. 3 is an aliquot part of several shipments of a sulphide ore of the low-lead class. This is the only known large occurrence of sulphides of this class in the district, though as an oxide it is the principal ore produced by several of the mines. The sulphur in the analysis more than satisfies the lead, zinc, and iron as galena, sphalerite, and pyrite, which is confirmed by the presence of these three minerals and the absence of pyrrhotite in all specimens of the ore examined. Arsenopyrite also occurs very sparingly, though not determined, and the gangue is composed principally of quartz and gypsum with minor amounts of mixed carbonates. The sample may also contain some oxides, as both sulphides and oxides are being mined, and in breaking the ore for shipment small mixtures are unavoidably made; but, except for the presence of the gypsum, this ore would fill most of the requirements of a sulphide of the silver-lead-zinc type, low-lead class. A large part of the deficiency in the total of the analyses is accounted for in the oxygen and water of the gypsum, which were not determined.

Analysis No. 4 is of a specimen of sulphide ore from one of the chimneys. Unfortunately, as is often the case with specimens, it runs unusually high in the most valuable metallic sulphide and low in gangue, but it is useful in conjunction with the others, as it so closely approximates a complete analysis. The mineralogic composition is pyrrhotite, small amounts of pyrite, a little oxide of iron due to exposure to the air, galena, sphalerite, and minor amounts of quartz and mixed carbonate.

No. 5 is an average of several samples of the deepest sulphide yet encountered, and is not far from being a normal average sulphide ore. Reconstructed, it is found to be composed of pyrrhotite, to the exclusion of pyrite, galena, and sphalerite, and a very little gangue. The sulphur in the average is a little low, but as this analysis was apparently made by rough commercial methods the discrepancy may be in the determination rather than in the ore.

These analyses are representative of the major part of the silver-lead-zinc sulphides now known in the district and considerable effort was expended to make them as reliable as possible; and while they leave much to

be desired, yet, used with care and discretion, they probably give as clear an idea of the primary ore of this type as can be obtained at the present time. It is to be hoped that in the future closer data may be obtained, especially with reference to the condition of the iron and the amount of sulphur in the unaltered ore. From these and similar analyses and a study of the records of several years' shipments of different mines, the following are offered as the probable analyses of the two classes of primary ores of the lead-silver-zinc type:

No.	Pb	Zn	Fe	SiO_2	S	Mixed Carbon-ates	As	Total
1, Per cent.	13	11	32	4	33	4	1.5	98.5
2, Per cent.	3	6	20	20	30	15	1.	95.

The first analysis, or that of the high-lead type, is probably approximately correct. The ore is practically a pure mass of dark-colored, medium-grained, slightly friable sulphides, with very little gangue, and with sphalerite, owing to its relatively low specific gravity, and consequent relatively large volume, predominate to the eye. The three principal sulphides, galena, sphalerite, and pyrrhotite, in some cases are very intimately mixed; while in others the pyrrhotite occurs in pure masses an inch or more in diameter and with the lead and zinc also segregated, but to a less marked degree. As far as could be ascertained, the pyrrhotite appears to have been the earliest mineral deposited, followed by the sphalerite and then the galena, although apparently the period of the sphalerite deposition overlapped and continued throughout the period of the deposition of the lead sulphide.

A recalculation of this analysis gives: Galena, 15; sphalerite, 16.4; pyrite, 23.5; pyrrhotite, 34.1; quartz, 4; mixed carbonates, 4; and arsenic, 1.5 per cent. The latter occurs as arsenopyrite, but for the purposes of comparison with the resulting oxide has not been recalculated. The iron sulphide is arbitrarily divided in the above recalculation. Development appears to have proved that the occurrence of the iron sulphide as pyrrhotite or pyrite depends almost entirely on the depth at which the ore is found, and, as nearly all the high-lead sulphide known at the present time occurs in the deepest portions of the mine, it contains pyrrhotite to the practical exclusion of pyrite. Thus four of the five analyses given above are of the sulphide ores that come from the deeper horizons, and carry pyrrhotite rather than pyrite, and hence run lower in sulphur than the theoretical analysis which is supposed to be an average, not only of the known sulphides, but also of the ores now occurring as oxides. In this connection, it must be taken into account that unmistakable traces of primary pyrite occur in the oxidized orebodies in the higher zones, and it seems probable that while at a greater depth a slight

modification of the above analysis may be necessary, for the present developed ground it is correct.

The low-lead analysis is not worthy of much weight, as there are few opportunities for obtaining accurate data with regard to this class of ore, sulphides of the low-lead class being a rare occurrence. The body sampled consisted essentially of pyrite—to the exclusion of pyrrhotite—quartz, minor amounts of galena and sphalerite, and considerable mixed carbonates. As with the other class of primary sulphides, the principal economic value lies in the silver content, which varies too widely to give averages.

All the silver-lead-zinc orebodies now known have probably resulted from the oxidation of one or the other of these two classes of sulphide ores, and intermediate types have not been encountered, though, where leaching and enrichment have taken place, local occurrences of oxidized ores are found, which are only with great difficulty correlated with either of these two classes. It is believed that the two classes of ore were formed at about the same period, though perhaps not exactly at the same time, and, if so, the low-lead class preceded the formation of the high-lead ores.

The general source of both classes, as will be shown later, is probably the same, but the specific sources, or better, the channels along which the two classes traveled in reaching their present locations, are in the writer's opinion entirely distinct. It is not believed that the low-lead ore is a phase or stage in the mineralization of a certain area by the high-lead class, but that if the two types are ever found intermingled, yielding an ore having the characteristics of both classes, this will be due to the impregnation of an orebody of one type by solution carrying the other type, each coming through distinct and widely separated channels. In looking over the occurrences of these two classes of ore, it is found that to date there has been very little overlapping of the areas characteristically yielding either of these types. The low-lead class is found in the northern part of the main zone and to the east; the high-lead class covers the south and central portions of the area.

All primary ores are believed to have ascended from below along the North-South group of fissures, selecting a favorable channel and replacing the limestone of the walls until a body more or less elliptical in cross-section, called a chimney, was formed. It is readily shown in many ways that the orebodies are not the result of the filling of open spaces, but rather of replacement of the limestone, as, for instance, chert nodules are found in the primary ore unreplaced and in their original positions. The North-South group of fissures, as above pointed out, are parallel and were probably due to the folding of the strata, and the most productive areas occur near the axes of the anticlines or a short way down either limb. Usually the chimneys are intersected by strong cross fissures, but it cannot

be shown that these have determined the location of the chimneys, as they are usually later than the stage of ore deposition, and are so numerous that it would be impossible to locate a chimney in the developed area without its intersecting one or more of these strong cross fissures. The exact cause of the localization of these chimneys is not clear, though apparently a favorable point for such an occurrence is the intersection of two fissures of the North-South group of slightly different trends, as a N. 30° W. and a N. 10° W. or a North-South and a N. 10° W. It is probable, however, that the true cause of the placing of the chimneys at the particular points along the North-South fissures at which they are found will only be determined when these orebodies are worked downward to their ultimate source, as will be considered later.

The solutions made off from these chimneys at favorable horizons, usually along the North-South group of fissures, replacing the limestone and forming the horizontal manta bodies, already briefly described. In breaking off from the chimneys, the bodies show their maximum irregularities, for they make their way through the limestone with apparently little regard for fissures or bedding planes; usually are inclined upward away from the chimneys and trend toward the structural axes, but at a distance of from 100 to 200 m. they pick up a favorable horizon and some North-South fissure and follow these for great distances. Some irregularities, however, are always encountered, the bodies crossing from one North-South fissure to a neighboring one, varying in total width and height, and stepping up or dropping down a few beds, though much of this vertical movement may be due to undetectable faults. On the whole, however, one is struck more by the continuity and regularity of these deposits than by the minor variations.

Some of the important points in this simple deposition of the primary ores are believed to be: First, that the solutions came from below the greatest depth yet reached; second, that they rose along North-South fissures only at certain favored points, forming chimneys, not along the whole or any considerable length of the fissures as has been sometimes considered; third, that the solutions broke away from the chimney, usually though not invariably upward and outward; fourth, that, encountering a favorable horizon and a North-South fissure, these were followed more closely than would be expected, considering the uniform character of the limestone and the multiplicity of the fissures.

As the most important corollary to the above propositions, the following is offered: No primary ore or ore oxidized in place can occur without a definite connection, through an orebody of reasonable cross-section, to the ore-bearing channels that go to a greater depth than has yet been reached. This is not an entirely accepted principle, as it has been believed by many that the ore found in the mantas rose, when in solution, more or less along the whole length of the fissure which the manta follows;

that the solution deposited its burden when a favorable horizon was reached, thus forming the manta; and that the chimneys resulted from the intersection of the ore-bearing fissure with some cross fissure. To sustain this view, fissures of the North-South group are occasionally found 18 in. to 2 ft. in width for considerable distances and over some vertical range, filled usually with oxidized ores which, excepting for a few important differences, occur as would be the case if the whole length of the fissure were the channel through which the ore entered the mantas. The objections to this idea are that enlarged fissures of all groups are found below the oxidized orebodies; that, when followed downward, they eventually close up completely and die out without connecting with any other channel that may have been the source of the ore; that the filling always shows ore leached in as distinguished from ore oxidized in place; and that even these occurrences are the exception and not the rule, as in general there are few signs of open fissures of the pre-mineral type below the manta bodies, it often being impossible to distinguish any North-South crack whatever in the limestone 100 ft. below a manta, even after the minutest examination. This is the case with several of the very biggest and strongest mantas now known in the district.

Were it possible, as has been sometimes considered, to have an orebody containing many thousand tons formed by deposition from a solution which traveled to this point along an almost microscopic crack, without leaving the slightest trace of mineralization to show that such ore-burdened solution had passed, then isolated bodies of ore could be found. It is the writer's contention, however, that it is not necessary to strain one's credulity and imagination to the acceptance of such a view, and that isolated bodies of primary ore or secondary ore oxidized in place do not occur in the Santa Eulalia district. In substantiation of this, nearly all known mantas can be traced decisively to the chimney which fed them, through continuous ore of normal cross-section, and where this has not been done, there is a field for development work.

The same principle is applicable to the chimneys to a less marked degree. While no one can say to what height above a certain point a chimney may extend, that is, how far upward it may make its way through the limestone, it is believed that true chimneys must extend downward beyond any depth yet reached toward the source of mineralization, perhaps eventually changing in composition and character, though little evidence of such a change has appeared.

The sulphide orebodies usually show a marked tendency toward horizontal bedding; owing to the replacement of the flat-lying limestone. This is particularly marked around the edges of the orebodies, where extensions, usually a few feet thick, are found making out into the walls on the more favorable beds, while in the chimney the unfavorable limestone beds sometimes lie nearly across the orebody, forming a false roof,

with a relatively restricted connection through them to the next favorable bed above. In an intermediate stage the slightly unfavorable beds are partly replaced by the ore, leaving large masses of pure limestone surrounded by the mineral matter. There is little gradation of ore into the limestone and the complete change from high-grade sulphide ore into almost pure limestone takes place within a few inches. Accompanying the sulphide ore in very rare cases are small open channels of circulation without mineralization, called dry caves, but these are clearly later than the ore and have not been found to date, except where the oxidation has commenced, and are apparently dependent upon the oxidizing solutions for their formation. Aside from these rare occurrences and a few vugs, measuring usually less than 6 in. across, open spaces visible to the eye are of rare occurrence in the sulphides of this type.

The Silver-Lead-Zinc Type Oxidized Ores

By far the greatest production has come from the oxidized ores of the silver-lead-zinc type; indeed, in comparison, all others are but interesting occurrences. To date, no unoxidized mantas of the silver-lead-zinc type have been found, so that the entire subject of these interesting manta bodies must be considered under the head of oxidized ores.

One of the things most desired from the practical standpoint of the development of new orebodies, and the following to ultimate ends of those bodies now known, is a clear, comprehensive knowledge of the process of oxidation in all its stages. Unfortunately, this is extremely difficult to obtain in the Santa Eulalia district. Our knowledge of the primary sulphides is limited; even the two theoretical analyses given above are open to criticism; and the changes which took place in the ore and the steps in the process of oxidation are poorly represented, the very slightly changed sulphide ores usually occurring in juxtaposition with completely oxidized carbonate ores, with the line of demarcation between the two sharply and distinctly drawn.

The only possible way in which this subject can be approached at present is to compare the normal sulphide with the normal oxide, leaving out the intermediate stages, and then consider the result of oxidation rather than the processes. The oxidized ores of the chimneys cannot be used in this comparison, for in the chimneys, as will be shown later, certain migrations of the metals have resulted in concentrations and impoverishments, so that, when the huge size and extent of these bodies are considered, the difficulty of arriving at the average character and content of the oxidized ore of the chimneys is at once apparent. In the mantas, however, other conditions obtain. The mantas lie practically horizontal, and, while the metallic content varies locally, the shipments of one month closely approximate the shipments of the year and these the shipments

of several years; and different mines on the same bodies, in spite of different methods of handling and mining ores, separating them into various products, etc., closely approximate each other. In fact, a compilation of yearly averages of several properties shows a variation of about 1 per cent. Pb, 1.5 per cent. SiO_2, 1 per cent. Fe, 0.5 per cent. CaO, 0.5 per cent. Zn, 1 per cent. S, and 0.5 per cent. As, though the silver, as the principal value, varies more widely. These come from one chimney and its mantas, and show a sufficient uniformity for comparison with the primary sulphides of a similar type and class.

In the following analyses of oxidized ores No. 1 is of the high-lead class and No. 2 of the low-lead class:

No.	Pb	Zn	Fe	Mn	SiO_2	CaO	S	As	Total
1, Per cent.	13.9	1.8	28.5	1.5	19.6	6.1	1.5	1	73.9
2, Per cent.	4.4	1.5	10	1.5	19	23.8	1	.....	61.2

Mineralogically recalculated No. 1 gives: Cerussite, 13.7; galena and anglesite (secondary), 3.8; smithsonite, 3.5; quartz, 19.6; sulphur, 0.5; gypsum, 2.7; calcite, 9.3; hematite, turgite, and limonite, 42.5; wad, 2.4; arsenic, 1; total, 99.0 per cent. The undetermined, besides the oxygen and carbon dioxide, are fluorine, chlorine, magnesium, barium, aluminum, phosphorus, antimony. Part of the lime should be figured as fluorite, though this occurs in almost negligible quantities; some of the iron should be with the calcite as an impurity; the form of the arsenic is indeterminate; but on the whole the recalculation is a very close approximation of the average composition.

The recalculation of the corresponding sulphides already given is: Galena, 15; sphalerite, 16.4; pyrite, 23.5; pyrrhotite, 34.1; quartz, 4; mixed carbonates, 4; arsenic, 1.5 per cent. (not recalculated). The theoretical specific gravity of the sulphide ore of this type and class is 4.8, while the true specific gravity of the ore masses more closely approximates 4.2, the difference being due to the porosity. The theoretical specific gravity of the oxidized ore of this type and class is 4.3, while the true specific gravity of the ore standing in the stopes is more nearly 2.4. The sulphide ore has then about 8 per cent. of void or open spaces, while the oxidized ore has about 44 per cent. of the same.

The number of tons of oxidized ore resulting from a given number of tons of sulphide ore is unknown, as are the relative volumes occupied by these ores, as will be shown later, so that there is but one basis on which a comparison of the two can be made, and that is on some element which remains constant in total in a given section. The lead apparently is the only metal which answers the purpose. It is not readily soluble, is relatively immobile, and if found outside of the walls of the orebody usually contains a gangue, which, averaged over a number of cases, closely approximates the general gangue of the ore; if it is partly removed by the

oxidizing solution from the original site of deposition, it is removed proportionately to the other elements, thus not interfering with the relative values obtained by the comparison of the two types.

Comparing the two analyses, we find that there is slightly more lead per ton in the oxidized ore than in the sulphide, hence the number of tons has decreased a very little during oxidation, and the proportion between the tonnage of the sulphide and the tonnage of the oxide is as 13.9 to 13. Applying this corrective factor to the comparison, we get the actual loss and gain per metric ton of primary sulphides during the oxidation process as follows: Loss, 93 kg. of zinc, 316 kg. of sulphur, 6 kg. of arsenic and 53 kg. of iron, making a total loss of 468 kg.; gain, 144 kg. of silica, 71 kg. of impure calcite and gypsum, 45 kg. of carbon dioxide, aside from that contained in the calcite, and 142 kg. of oxygen, making a total gain of 402 kg. The resulting orebody contains almost the same number of tons of ore as did the primary body, but the secondary ore, owing to its lower specific gravity, occupies a much larger space. While the above is open to criticism in the analyses themselves, still it is believed, not only that it is a close approximation of the truth, but also that in no other way can a conception be gained of the actual results of the oxidation process.

If it were possible to obtain an analysis covering an equally large amount of the low-lead class of ore, this could be treated in the same way, but, unfortunately, both the sulphide and oxide analyses given for this class of ore have not sufficient weight to warrant mathematical deductions.

The oxidized ores, particularly the high-lead class, are predominately soft, friable masses of siliceous iron oxides and lead carbonates, loosely massed together and extremely porous. Though all possible variations and combinations occur, a typical cross-section of an oxidized manta orebody is about as follows: The roof is relatively flat at the center, being nearly always a bedding plane of the limestone; 40 or 50 ft. below, the floor is rounded, with perhaps a rapidly diminishing extension downward along some fissure; the walls are 100 to 150 ft. apart, and one or both are often irregular, with mantas 5 or 6 ft. thick extending out into the limestone a few meters. On the roof is a light yellowish brown, extremely finely divided earthy deposit several inches thick, composed principally of silica and alumina with a little magnesium carbonate, manganese, and iron oxide. Below this is an open space varying from a few inches to 10 or 15 ft.—in an orebody with the dimensions given, being often 4 or 5 ft. deep—and extending clear across the orebody. This open space may often contain beautiful growths of calcite, stalactites and stalagmites, and fine branching intergrowths of calcite tubes, varying from 1 to 1/64 in. in diameter, with those under 1/4 in. in diameter hollow and built up by lime-bearing water traveling along the interior. In many cases, however, these calcite crystals and growths are lacking.

Below is the surface of the ore, sometimes with a few boulders of limestone fallen from the roof, making an irregular surface, on which is almost invariably a coating several inches thick of gypsum which can best be described as closely resembling snow. It is as soft as snow, has a slight crust easily broken, can be pressed into balls in the hands, and lies on flat or gently sloping surfaces, and not on steep ones. Sometimes a little wad or pyrolusite is found in places lying above the gypsum, while below is often a layer of from 2 to 6 ft. of nearly pure cerussite or lead carbonate, bluish in color and so soft as to be readily shoveled without picking. From this point downward, the ore is a mixture of siliceous iron oxide and lead carbonate, lying loosely on the limestone floor. Along the walls are found the little galena and anglesite that occur with the ore, and where they are present in considerable amounts these secondary sulphides extend out into the walls as solid mantas several feet thick.

In the vertical chimneys of the high-lead class portions containing little lead are common, while just above the sulphides at one point a mass of pure cerussite 50 ft. thick and the full size of the chimney was found; a result of concentration of values in these vertical bodies. Large open spaces are common in these oxidized chimneys, and where an unreplaced bed of limestone extends nearly across the body an open space occurs just below it, similar to that found below the limestone roof of the manta bodies.

These open spaces have been attributed to "shrinkage," on the supposition that, due to the loss of sulphur and zinc, the ores must have lost in bulk during the oxidation, and this term certainly describes the appearance of the open spaces admirably. However, in the consideration of the results of oxidation it was shown that the decrease in a total number of tons is so slight as to be almost negligible, and that, owing to the lowering of the specific gravity of the oxides and the increase in porosity, the oxide ore occupies a greater space than the sulphides, the ratio being inversely as their specific gravity or as 42 to 24. Locally, portions of the entire orebody have been removed into cross fissures. These figures are based on the ore alone and include porosity, but do not include the open spaces (or "shrinkage") which occur in the oxides but not in the sulphides. This factor, then, further increases the size of the carbonate orebody, so that its cross-section from limestone to limestone must be increased during the oxidation process at least one-third in all cases and probably often one-half. This accounts for the discrepancy in size between the oxides and sulphides noted in some of the chimneys, for the cross-section of the sulphide or deeper portion is very much smaller than the cross-section of the oxidized chimney above, which, under the theory of "shrinkage" of carbonate ores, caused some apprehension as to the prospects of the district with deeper mining. Such apprehension, however, is falsely founded, for, when carefully considered, it is found that the tonnage remains the

same, if anything slightly increasing with depth, it being only the distance between the limestone walls that has decreased with the change in character from oxides to sulphides.

It is apparent that such increase of size of the ore-bearing cavity must be due to the solution of the limestone wall rock of the orebody during oxidation; and a consideration of the probable process of oxidation shows that undoubtedly such action has taken place. The sulphide with its four principal metallic minerals, pyrrhotite, pyrite, galena, and sphalerite, is acted upon by oxygen-bearing waters which also at some stage carry considerable silica. The iron sulphides were first attacked, probably yielding ferrous sulphate and sulphuric acid; the former then passed to the hydrous oxide with the generation of more sulphuric acid, and very probably with the intermediate step of the formation of the ferric sulphate, itself known to be a strong oxidizing agent. The sphalerite is readily attacked by these iron salts and the free sulphuric acid, but, although its great mobility and ease of solution are well shown at Santa Eulalia, it apparently does not commence to oxidize until the oxidation of the iron sulphides is well under way. It certainly formed zinc sulphate as a stage of the oxidation, but goslarite is found only sparingly as an efflorescence on the limestone, while the usual deposit along channels of circulating waters is the carbonate, and it seems probable that much of the zinc was carried away from the orebody in solution as zinc carbonate, after the reaction between the zinc sulphate and the calcium carbonate had resulted in gypsum and smithsonite. Hydrozincite is not rare at Santa Eulalia, but its relation to the process of oxidation could not be determined. The oxidation of the lead follows slowly and there is abundant evidence that the galena is attacked strongly only when intimately mixed with the other more readily oxidized sulphides, due probably to the generation of sulphuric acid and sulphates of iron in the mass of mixed ore, and to the removal of the more soluble sphalerite, with the consequent development of a cellular porous structure in the galena. When recrystallized as secondary galena, in which form a small amount of lead occurs in all orebodies, it is free from iron and zinc sulphides, and in this pure state is almost stable. It then apparently undergoes oxidation with extreme slowness; much of it remains unchanged, but a fair proportion goes over into the sulphate, anglesite, and this also, when pure, appears to be practically stable, yielding only a slight coating of the carbonate, cerussite. This marked difference in solubility between the primary and secondary galena appears to be due, first, to the differences in physical character, the primary being cellular, while the secondary, being massive, is only attacked on its exterior surfaces; and second, to the fact that the main processes of oxidation having been completed, the generation of sulphuric acid and the formation of active ferric sulphates practically ceased before the reoxidation of the galena commenced. The sulphate, anglesite, may

be deposited in considerable quantities for the first time during the oxidation of the secondary galena, for, although the formation of the lead sulphate is believed to be a step in the alteration of the galena to cerussite, it is not commonly found in Santa Eulalia as an alteration product of the primary ore. Apparently the reaction between the lead sulphate and the calcium carbonate took place immediately upon the formation of the sulphate, and the process was recorded only by the deposition of the cerussite. This same stability is noticed in the third or last stage of primary mineralization, the narrow galena seams extending up through the porphyry to the surface. Although these seams outcrop, they are unaltered.

In many of these processes free sulphuric acid is liberated, at least a portion of which probably attacks the limestone walls of the orebody, forming gypsum and enlarging the sides of the chamber, while the sulphate radical acts similarly in the double decomposition of the lead and zinc sulphates with the calcium carbonate. A comparatively insignificant part of the gypsum thus formed remains as a coating on the orebody, as above noted in the description of the typical cross-section of the average manta, but by far the greater part of it is carried off with the uncombined sulphuric acid in the circulating waters. The insoluble portions of the limestone, usually a very small percentage of silica and a little clay, remain as a finely divided deposit in part on walls and roof, but principally in the deepest parts of the floor of the orebody along the channels of circulation.

Thus the orebody, which as a sulphide was strictly a replacement of the limestone, as an oxide takes on the characteristics of a deposit formed in a solution chamber.

During the oxidation, a certain amount of readjustment of the metals remaining in the orebody took place. Apparently the principal reaction between the solutions and the limestone occurred along the roof of the orebody, as shown by the bed of pure cerussite, just under the gypsum, and by the residual clay deposited against the roof; considerable action took place along the walls also, where the secondary galena and accompanying anglesite are found almost exclusively.

In general there is abundant evidence that the action of the oxidizing solutions following the open channels has been downward, and indications are found in the water channels that the more migratory minerals, smithsonite, gypsum, etc., passed along these channels in that direction. Mineralization occurs in cross fissures underlying the orebodies from which it was derived, and with which it is often connected, and while iron and manganese stains are the rule below the oxidized orebodies, fresh unstained limestone is the rule above. These observations apply to the mantas of both classes rather than to the chimneys, for the latter offer open channels for circulating waters.

There is in the manta bodies, moreover, no secondary enrichment in the sense ordinarily used in deposits in veins where minerals are leached out of upper portions and reprecipitated in lower portions of the same vein, because first, in the main zone the average oxidized manta occurs at a depth of about 800 ft., very few mantas are found even locally at less than 200 ft., and the capping itself is often 400 or 500 ft. thick; second, the solutions that pass through one manta body do not necessarily encounter any other orebody in their downward course; third, silver, the principal economic mineral, seems to remain with the lead and hence usually stays in the orebody; fourth, no definite water level has yet been encountered, although several shafts have found a very small flow at about 1,500 ft.

A typical example of an orebody in a cross fissure or an orebody extending downward along the main fissure, due to the action of oxidizing solution, shows: First, the normal orebody or source; then, usually narrowing rapidly, practically normal ore carrying lead carbonate; below this, iron oxide carrying no lead; zinc carbonate grading into a mixture of fine siliceous clay and hydrozincite; and then a tight fissure. One such occurrence measures an average of 40 ft. across, and each of the stages shows nearly 100 ft. of vertical range. It will be noted that these minerals occur in the normal order of their solubility, the most soluble traveling farthest from their source and the finely divided clay, probably carried practically in suspension, collecting at the bottom. Such an ore occurrence is nearly complete in itself; that is, every constituent but a portion of the calcium carbonate and the gypsum being accounted for. The fissure, therefore, offers no inducement to prospecting downward except the possibility of discovering another independent body. At laest one such orebody is known in each of the principal mines of the camp; nevertheless this is not the usual occurrence, for normally there is no obstruction in the channels to oxidizing solutions and the more soluble products are carried beyond the range of observation and cannot be completely accounted for.

As already shown, there was a loss, during oxidation, of 93 kg. of zinc per ton of primary ore, which is approximately a ton of carbonate ore; and when the total production of the carbonate ores to date is considered it is at once apparent that the circulating oxidizing solutions have handled an enormous tonnage of oxidized zinc minerals. To date, with the exception of such minor occurrences in cross fissures and the extension of orebodies downward as mentioned above, very few zinc bodies have been found, and the tonnage produced from these amounts is but a fraction of that yet to be found. As compared with other districts, the oxidation in Santa Eulalia was characterized by plentiful amounts of water, usually, though not always, circulating with comparative freedom, and by a deep water level not yet reached. In deposits in limestone, where

the water is largely confined to the orebodies, the smithsonite and calamine resulting from the oxidation are often found in the orebodies themselves, deposited along the walls and especially along the floors of the mantas, runs, blankets, flats, or whatever they may be called. Where the oxidation has been effected by barely perceptible trickling waters in a dry mine, zinc is often found in the fissures and caverns in the neighboring limestone, in crusts and veinlets sometimes but a few inches thick deposited directly from the supersaturated solution, as smithsonite and calamine; but these minor occurrences in the aggregate account for the total zinc of the unoxidized ore bodies. Where the amount of water was great and the circulation was free, as was evidently the case in Santa Eulalia, the zinc minerals were carried far from their source, and the channel through which the unsaturated solution passed may not contain even a trace of the oxidized minerals, and may simply show that a large amount of solution of some sort has passed through it.

The problem is not a simple one, owing to the multiplicity of channels in the limestone; but in some cases it has not received the attention which the value of possible positive results should warrant. It is not easy to recognize the zinc minerals resulting from oxidation, smithsonite alone having an almost endless variety of forms, many of which are extremely inconspicuous, and it is probable that only by continued systematic sampling for zinc can the problem be attacked with any degree of certainty.

The plotting of such zinc assays gives some interesting results. On one of the large mantas of the district this was attempted and it was found that a curve could be drawn which showed by its variation the location of the channels of circulation, for where the manta passed over a downward extension of the ore the zinc content was low, rising with increasing distance from this point and falling again where another downward extension or an open fissure was passed. If these downward extensions are true chimneys, the zinc content does not rise in them until the level of the sulphides is approached, at which point a body of high-grade zinc ore sometimes occurs in the limestone near the chimneys. On the other hand, if the extension downward is merely a local enlargement, the zinc rapidly increases with depth, and the clay content increases, under which conditions further exploration downward is inadvisable at that point. Thus the recording of zinc assays not only assists in the locating of bodies of zinc ore, but also shows clearly the channels of circulation of the oxidizing waters and those points at which prospecting should be done to determine the downward extension of the orebodies.

As will be readily realized, these conditions are capable of endless variation and the data must be carefully taken, thoroughly studied, and correctly interpreted, in order that a sufficient benefit may be derived to repay the labor involved in its compilation.

One of the best indications that a water course, channel, or fissure has been the locus of oxidizing waters that have acted upon and assisted in oxidizing an orebody, is the presence of a deposit of gypsum along it. This mineral also travels great distances from its source and is deposited in the limestone in caves or caverns, lined with beautiful white crystalline deposits of the mineral, and is sometimes spread out as an impregnation of the limestone itself. Such fissures or channels are favorable sites for the deposition of zinc carbonate or silicate.

"Painted limestone," or limestone stained with dark manganese oxide and pink carbonate and silicate, has long been recognized as a favorable indication of the vicinity of oxidized ore. Indeed, in a few instances it carries sufficient silver to be an exceedingly valuable ore in itself, and one highly prized by the smelter. Such ores occur, first, where the orebodies are relatively high in silver; second, where there is a special inducement to impregnation of the limestone, as, for example, where a chimney has jumped from one fissure to another, leaving a vertical oxidized porous chimney of ore practically resting on limestone; third, where the limestone for physical reasons is favorable to impregnation by descending solutions. In general, manganese-stained limestone is not necessarily silver bearing, and occurs to some extent along all fissures which have carried products of oxidation, except where these are encountered at great distances from the source of the oxides and consequently the solutions have been divested of all but the most soluble minerals. Gypsum and zinc compounds are thus found along a fissure or channel long after the manganese has ceased to be present. Normally the manganese-stained limestone occurs beneath the entire orebodies, never above, excepting along some fissure that carries oxidation products of some other orebody still above, or along the channel of the deposition of the third or pure galena type of mineralization.

The amount and intensity of this "painting" is an indication of the distance from the ore above. Were it not for a few drawbacks, this "painting" would be an invaluable aid to prospecting for oxidized manta bodies. As it is, it is merely a helpful indication to be carefully applied. The "painting" is often noticeably lacking, particularly in the vicinity of strongly marked open cross fissures, where the solutions have followed these open channels instead of impregnating the limestone mass. Ore may thus be found overlying unstained limestone. Certain horizons in the district, unusually favorable for deposition and in which several orebodies have been found, are "painted" by manganese carbonates and oxides over great areas, and it is then a question whether impregnation took place during the primary deposition, during the oxidation, or whether minute traces of manganese were deposited coincidentally with the limestone itself. Very often this type of "painting" occurs immediately below beds of fossils, and, while not continuous over the whole district, it

certainly is much more widely distributed than the orebodies. It has never been found associated with the sulphide ores and, if present below the level of oxidation, it is either very inconspicuous or the proper horizon has not yet been encountered. The great extent of this class of "painting" tends to mislead rather than to assist in the development and location of orebodies.

The origin and source of the large amounts of silica added during oxidation to all known occurrences of the high-lead class of ores, and to some occurrences of the low-lead class, are not clear, nor do they appear readily determinable. The channels in the limestone along which the oxidizing waters have passed are not silicified, nor are any large occurrences of silicified limestone known in the district. The capping, however, has been silicified over extensive areas and it is but natural to conclude that the same solutions effected the silicification of the orebodies. The most illuminating occurrence is that found in one of the deposits of the low-lead class, where the chimney stands for nearly 800 ft., running 20 to 30 per cent. higher in silica than the normal primary ore of this class and than some of the mantas. Where the chimney joins the enormous upper mantas the highly siliceous ore appears to be present in chutes, conveying strongly the impression that it was introduced after the period of oxidation, and that it followed channels through the ore rather than penetrating the whole mass. It seems probable, from our present limited knowledge, that the silica was introduced at about the same period as the third or pure galena type of primary ore deposition, but whether simultaneously, just before, or just after, is not certain. It is probable that it came from below; that it followed the oxidized chimney as a channel; that it continued upward into the overlying capping; and that it represented a distinct stage which has now passed.

The oxidized ores present an unlimited number of interesting occurrences, each illustrating some point or factor in the study of these deposits and each requiring separate investigation and a separate description, but the general principles governing the deposition are believed to be essentially those given.

"Contact" Bodies.—One of the points on which there is much difference of opinion among the operators of the Santa Eulalia district is the relative ages of the mineralization and the tuff, and the value of the contact as a locus of orebodies. The latest phase of primary mineralization, the third or pure galena type, resulted, as already described, in narrow vein-like streaks of galena, free from iron sulphide or sphalerite, usually about an inch in width. These narrow galena seams are found throughout the limestone mass, but only in the upper horizons thus far. They extend up into the overlying beds of tuff, usually on N. 30° E. fissures, thus showing that this tuff, where fissured, is capable of mineralization and that the capping was present when this last primary action was taking

place. Had the capping been present when the main silver-lead-zinc ores and silver-bearing pyrite ores were being deposited, traces of these mineralizations would probably be found in the capping.

The dikes cut both the orebodies and the capping and, as has been pointed out, were later than the primary ore and earlier than the oxidation. Hence the capping was present during practically all of the great period of oxidation and its accompanying ore movements and was a considerable factor in controlling the circulation of the oxidizing waters.

The relative ages of the tuff and the mineralization cannot be absolutely determined by the study of the various fissures and their ages, but the results are not without interest in this connection. No definitely certain pre-mineral fissures continue upward into the overlying tuff and, of the earlier post-mineral fissures, only those that have strongly marked later fault movements do so, while the latest fissures outcrop characteristically on the surface of the tuff, often having marked effect on the topography and sometimes, as noted, carrying mineralization of the third or pure galena type into it. It is conceivable that fractures may have occurred in the limestone, continued up to the tuff, and died out without entering even a few inches into the overlying rock, but it does not appear probable; and it seems rather that the study of the fissures in this connection places the age of the tuff as later than the pre-mineral fissures, later than the first post-mineral cross fissures, and earlier than the last post-mineral cross fissures. Hence both of the economically important types of ore deposition appear to antedate the capping.

The oldest type of silver-bearing sulphide ore occurs directly under the tuff at one point where the ancient surface was low, and it is scarcely conceivable that this type of mineralization could take place at the shallow depths (probably at no time exceeding 1,500 ft.) represented by the overlying capping. Certainly, if this were true, a reconstruction of one of the basic ideas of economic geology—that the minerals are a criterion of the depth at which deposition took place—would be necessary. For the same reason it does not seem possible that the lead-silver-zinc bodies of such great vertical range could have formed at such a relatively shallow depth. (See p. 97.)

If the tuff had been present when the ore was rising along the chimney, action resembling that found in similar deposits in a region where shale is interbedded with the limestone or overlies it would be expected; that is, there would be a widening out or spreading of the deposit just below the impervious beds, particularly developed where the shale or tuff was practically horizontal, or better, where it formed a large dome. Such occurrences are common in the lead-silver-zinc deposits in many districts, but nothing of a similar nature occurs at the contact between the tuff and the limestone in Santa Eulalia.

The known occurrences in the main zone at Santa Eulalia are illus-

trated by three orebodies partly or completely worked. These bodies are all similar in that they are rather small and erratic, yielding but a small fraction of the ore produced from the average manta, and this is often concentrated in caves or bodies with relatively little ore between. In no case do they follow a North-South fissure or any other system of fissures for any great distance, but break from one set to another, apparently following the most promising channel, regardless of direction. The contact is in each case steeply inclined, and the action, where it could be determined, appears to have been downward along the course of the fissures rather than upward. There is no tendency to spread out along the contact; in fact, the ore very often is not in contact with the tuff, but from 5 to 30 ft. below, in the limestone, though always touching the sloping contact at intervals. Occasionally some impregnation occurs in the tuff, but this, as nearly as can be ascertained, took place as an oxide, not as a sulphide. The ore itself varies considerably; one of these bodies belongs to the low-lead type, the other three to the high-lead class, but only in one of them was the lead present in appreciable quantities.

It seems probable that these bodies were deposited in their present location as oxides, the ore-bearing solutions working downward along the most favorable channel, in this case that formed by the intersection of fissures of all ages with the contact itself; and that the source of the ore was in all cases manta orebodies which lay normally in the limestone, just as the known orebodies lie to-day, but which outcropped on the old surface before the tuff was laid down, just as several mantas outcrop in the cañons of the present topography.

The mantas lay practically horizontal, as sulphides, while the erosion was going on, the climate of that time in Mexico undoubtedly being such that erosion took place as rapidly as oxidation. Certain of the orebodies were exposed on the surface of the steep hillsides, particularly on those slopes running approximately east and west, or at right angles with the general north-south trend of the orebodies, while in other cases they were completely removed, as evidenced by the indications on the surface at certain points. Those bodies exposed on the hillside and running back into the limestone were buried again under the showers of tuff or volcanic ash and were later oxidized. As the whole cross-section of the orebodies was exposed at the sloping contact or old surface, a natural channel for circulating waters, it is not remarkable that great quantities of oxidized ore were carried down along this channel and deposited in it. Two of the known contact bodies are believed to have resulted from one of these mantas since completely removed by erosion, and in consequence the contact bodies themselves now outcrop on the surface. In other cases the contact bodies, if followed upward, would undoubtedly lead to a manta in the limestone at a higher horizon, and a case is conceivable

where, after following such a manta for some distance, the contact would be encountered again and a second contact body similar to the first be found pitching down the other slope.

There is one case in the district where a strong manta following definitely a North-South fissure, lying at a short distance below the contact and in places touching it, has all the characteristics of a typical manta orebody, in size, in cross-section, oxidation products, etc.; it lies practically horizontal and follows a certain bed of limestone that is apparently favorable for deposition of ore. This is not a contact body and the fact that the contact is sometimes just above and always within 100 ft. has nothing to do with its occurrence; it seems more probable that the favorable limestone lying just below the tuff, which is associated with it, is the determining factor. In substantiation of this view, other bodies are known in Santa Eulalia which lie for some distance at a scarcely greater depth below the present surface; given a slightly longer period of erosion and another deposition of tuff, the same sort of occurrence as that just described would result.

It is believed, then, that the tuff was laid down after the principal period of primary mineralization, and certainly after all primary mineralization of economic importance; that it antedates the great period of oxidation and accompanying readjustment; and that the limestone just below the tuff is as favorable for oxidized manta orebodies as other limestone horizons, but not more so, unless for some reason independent of any consideration of the overlying tuff. Also, that the contact, where steeply pitching to the north or south, is a favorable location for minor occurrences of secondary ores, which are economically important principally as they may lead upward into the oxidized manta at a higher horizon of the limestone.

Probable Source of the Ores

We have, thus far, in a brief and incomplete manner considered the primary deposition of the three types of ores, their occurrence and their process of oxidation. There remains only the consideration of their ultimate source, a subject dealing purely with horizons not yet reached in Santa Eulalia, and conditions that can be only conjectured.

It is evident that at Santa Eulalia the two economically important types are far more distinct than is usually the case with the variations found in a single restricted area and a single formation. The silver-bearing iron sulphide type is distinct from any other known deposits in its mineralogical association of ilvaite, fayalite, knebelite and magnetite as gangue minerals, and silver-bearing pyrrhotite and pyrite and arsenopyrite as metallic sulphides. The development of the type is not complete and it is very possible that with the present 1,000 ft. of vertical range more fully exposed, conclusive evidence may be obtained as to

the manner of its formation. It undoubtedly belongs to the class of deposits illustrated by the Ducktown, Tenn., and Granby, B. C., copper mines; that is, deposits resulting from igneous metamorphism from a source not yet exposed. The relation of the feeder fissures and the shape and occurrence of the orebodies seem to point toward a source lying vertically below.

The later type of silver-lead-zinc ore at Santa Eulalia is remarkable for its great vertical range, having been traced through 2,000 ft. of limestone without appreciable change in metallic content. It is usual, in deposits of this type, to find the zinc and particularly the iron sulphide increasing with depth, resulting in an ore that resembles the metallic sulphides of the silver-bearing iron sulphide ore of Santa Eulalia, but to date such a change is not apparent.

These characteristics would seem to point to the formation of both types of deposits at unusually great depth and to an exceedingly protracted period of mineralization during a very slowly falling temperature. Lindgren points out that the silver-lead-zinc type may be deposited at depths exceeding 12,000 ft. (*Mineral Deposits*, p. 513 [1913]), and the gangue minerals of the silver-bearing iron sulphide type of Santa Eulalia are normally higher-temperature minerals than the silver-lead-zinc type. It seems probable, then, that the deposits at Santa Eulalia were formed at least 10,000 ft. below the surface and possibly at much greater depth.

Approaching the subject from a consideration of other mineral occurrences lying in the same Cretaceous formation and in the same metallographic province, some additional ideas are derived. J. E. Spurr and G. H. Garrey have studied two of these deposits in great detail,[5] and from these and similar occurrences Spurr was able to arrange the widely varying types of mineralization in a column showing that all these represent progressive changes in the mineralizer which has its source well within the igneous rock, and that this mineralizer is essentially but a magmatic phase. He has given[6] seven zones, or stages, arranged progressively, of which stage C—cupriferous pyrite, stage D—argentiferous pyrite and auriferous arsenopyrite, stage E—zinc blende, and stage F—argentiferous galena, are extremely common in northern and north central Mexico, throughout the Cretaceous limestone. The writer has had on opportunity to study in some detail both occurrences described by Spurr and Garrey, since the appearance of these articles, as well as numerous other deposits in the same metallographic province, and, while probably some slight changes in the grouping or subdivision may be necessary in the

[5] J. E. Spurr, G. H. Garrey and Clarence N. Fenner: *Economic Geology*, vol. vii, No. 5, pp. 444 to 484 (Aug., 1912).

J. E. Spurr and G. H. Garrey: *Economic Geology*, vol. iii, No. 8, pp. 688 to 725 (Dec., 1908).

[6] *Economic Geology*, vol. vii, No. 5, p. 489 (Aug., 1912).

future as they become better exposed by further development, the general idea admirably fits the ore occurrences of this section. One of the difficulties that stand in the way of its ready application to practical problems is that the arrangement was necessarily made according to two factors rather than a single one: first, according to the physical conditions, or the decreasing temperature and pressure as the distance from the seat of igneous activity increased and the distance from the surface decreased; and second, according to the time of deposition or the gradual changing of the mineralizer and its metallic contents as time progressed, independently of the temperature. Either of these factors may cease in its normal progression; for instance, the recurrence of igneous activity after any number of stages have been passed may start afresh the entire series, or intense faulting, or the extrusion, as dikes or flows, of large masses of igneous rock may at any stage in the process so reduce the physical conditions of temperature and pressure over local or extensive areas as to allow the deposition of the remaining stages to be superimposed over the earlier. Thus one case has been noted where a fault of great magnitude occurred after the conclusion of the "C" or cupriferous pyrite stage, and bodies of the "E" and "F" or sphalerite and galena stages were found largely along the fault zone and slightly impregnating the orebody, it being as yet unknown whether the deposition of the "D" or argentiferous pyrite and auriferous arsenopyrite stage took place before or after the faulting, though this knowledge is of primary practical importance, as in the first case stage "D" would lie above the copper stage and has then been removed by erosion, and in the second case it would lie below the sphalerite and galena stages and hence must be looked for with depth.

At Santa Eulalia we have exposed the "F" or argentiferous galena zone in the third or latest type of mineralization, though only as an unimportant mineral occurrence; "F" and "E," or lead and zinc stages, strongly overlapping, as these stages often occur in the silver-lead-zinc type or main ore occurrence; and "D," the new, more recently discovered silver-bearing iron sulphide type, containing argentiferous pyrrhotite and pyrite and slightly auriferous arsenopyrite, and carrying as a gangue iron-bearing contact-metamorphic minerals belonging to a still deeper stage. For some reason, similar to those just outlined, the stages overlap. It is probable that as depth increases we will find increasing amounts of metamorphic minerals, these changing gradually to the deeper-seated types; and with this change will come, possibly, increasing amounts of copper-bearing sulphides, and finally the igneous rock, probably a monzonite or one of similar composition. When this rock is encountered and explored, we will probably find that the cause of the location of the silver-lead-zinc chimneys is due to irregularities in the shape and occurrence of the igneous rock rather than to intersecting fissures, etc. The depth at which this igneous rock may be encountered cannot even be estimated. Spurr found at Dolores nearly a mile of vertical range covering only

portions of the argentiferous pyrite-auriferous arsenopyrite zone and the cupriferous pyrite zone, showing that under favorable conditions one of these stages may have a great vertical range. It seems probable that at Santa Eulalia, as the lime-iron silicates have already appeared, the distance to the igneous rock cannot be great; still the thickness of the limestone is unknown and the igneous rock will probably not be a factor in the development of the district for many years.

Summary

In the following chronological table, the principal events in the formation of the Santa Eulalia ore deposits are arranged as they appear to have taken place.

1. Limestone laid down in Cretaceous sea.
2. Land surface uplifted at close of Cretaceous, with little development of structure.
3. Hypothetical igneous mass intruded; slight structure in limestone developed.
4. North-South class of North-South group of fissures formed.
5. Magmatic processes set up at some point in the mass of igneous rock, accompanied by the giving off of mineralizers which formed hypothetical contact-metamorphic deposits at some depth below that yet reached.
6. Silver-bearing iron sulphide type of ore deposited along North-South fissures.
7. Other classes of the North-South group of fissures formed.
8. Silver-lead-zinc type deposited.
9. N. 55° W. fissures formed.
10. N. 70° W. fissures formed, followed by a long period of active erosion of the limestone surface perhaps amounting to 10,000 ft. or more.
11. Capping of tuffs and volcanics laid down.
12. Andesite and rhyolite dikes intruded.
13. N. 30° E. fissures formed.
14. Oxidation on a large scale commenced.
15. Silicification of much of the oxidized ore and overlying capping took place, with mineralization of the third, or pure galena, type just preceding or following it.

Acknowledgments

The writer is indebted to the Managers and Superintendents and staffs of the mines of the district for many kindnesses, every opportunity having been afforded him for the gathering of the data upon which the paper is based, and wishes particularly to acknowledge his indebtedness to A. L. Eaton, Superintendent of the Chihuahua and Potosi mining companies, for much helpful information.

The Cloncurry Copper District, Queensland

BY W. H. CORBOULD, CLONCURRY, QUEENSLAND, AUSTRALIA

(New York Meeting, February, 1915)

THE township of Cloncurry is situated in the northwestern part of Queensland, about latitude S. 20° 42′ 53″ and longitude E. 140° 30′ 25″. Townsville is the port through which all the trade passes and is about 470 miles east-northeast of Cloncurry township.

The Cloncurry copper district covers an area about 200 miles long by 40 miles wide. Unfortunately the mines are very scattered over this area, there being no large groups of mines.

Mount Elliott mine is situated about 70 miles south from the township of Cloncurry, and Mount Oxide mine about 150 miles north of Cloncurry.

The district has been known for the last 35 years, and in the early stages a large amount of capital was expended with little to show for it; but within the last four years the railway has been extended from Townsville to Mount Elliott and about 40 miles north of Cloncurry, which has made possible the treatment of the ores of the mines on this extension.

The Mount Elliott company within the last few years has smelted and converted on its mine 169,688 tons of ore for 20,554 tons of blister copper, containing 20,377 tons of copper, 35,642 oz. of gold and 28,841 oz. of silver, at a cost of about £33 per ton of blister copper, f.o.b. Townsville. This cost included all development, wages, salaries, general management, office, agency, legal expenses, freight, etc. The gold and silver contents would practically pay for freight to Europe, refining, realization charges, etc.

The costs of supplies are as follows: Firewood, 15s., coke, 75s., mine timber, 54s., and coal, 60s. per long ton; explosives: dynamite and gelignite, 50s., and blasting gelatine, 65s. 6d. per case.

Mount Elliott Mine

The pay shoot of the orebody outcrops at the surface for about 200 ft., and lies between two slate walls about 40 ft. apart, having a general

underlie down to the 180-ft. level of 20° from the vertical; below that depth the ore channel widens out to represent an inverted funnel. The general strike is northwest at the surface, and corkscrews with depth until at the 500-ft. level the strike is nearly due west. The dip follows the strike, and from the surface to the present deepest level is at an angle of approximately 15° to 20°. The copper occurs in oxidized form from the capping down to the 230-ft. level, where the zone of secondary enrichment is encountered, in which the copper is in the form of friable, altered or sub-sulphides. This zone is from 50 to 180 ft. wide, overlying

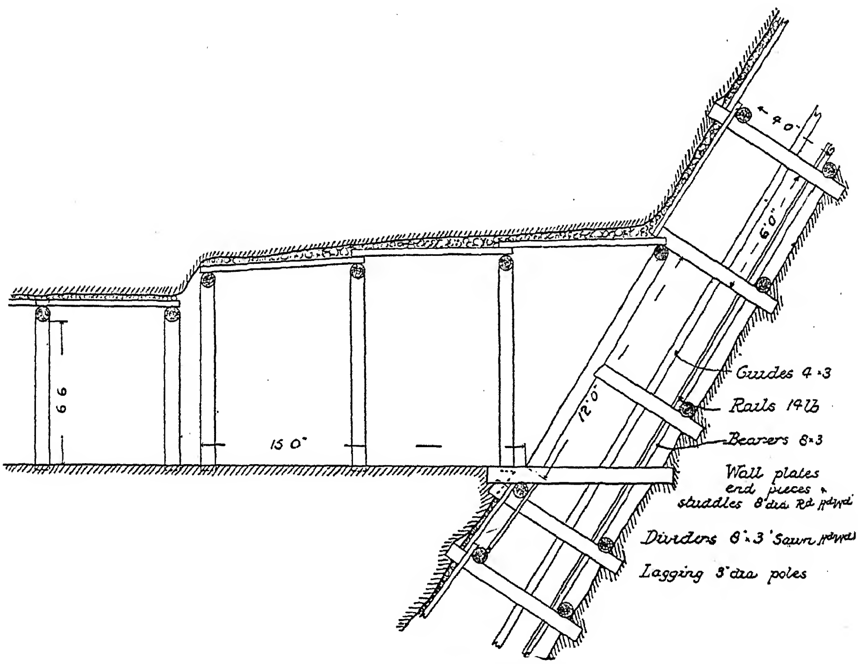

Fig. 1.—Main Shaft at Mount Elliott Mine.

the primary ores, which consist chiefly of chalcopyrites in hornstone. These continue in depth to the present deepest level, 600 ft. from surface.

The main hauling shaft (Fig. 1) is sunk in the foot-wall country at an inclination of 30° from the vertical, and is divided into three compartments, two hauling and one ladder way.

The shaft is secured with frame sets cut from 8-in. round timber, spaced 6-ft. centers, and lagged with small poles about 3 in. in diameter.

Each hauling compartment is laid with a track of 14-lb. rails, 18-in. gauge, on 8 by 3 in. continuous bearers of sawn hardwood. The rails are fish plated, and spiked to the bearers with 3-in. "dog spikes."

Levels are opened at 120, 180, 280, 400, and 500 ft. vertical depth,

and stations cut, from which the main crosscuts go out to the orebody. (The outcrop was mined by opencutting to a depth of 60 ft.) Drives were put through the orebody in the Nos. 1 and 2 levels for the full length of the pay shoot, with crosscuts to both walls at regular intervals, the main levels being subsequently driven in the foot-wall country.

At the No. 3 level the channel, which has an area of 180 by 150 ft., was contoured, the drives being kept just on the fringe of the lode material, with intervening drives and crosscuts dividing the area into convenient blocks for stoping.

All drives are timbered with sets and covered with split slabs 6 in.

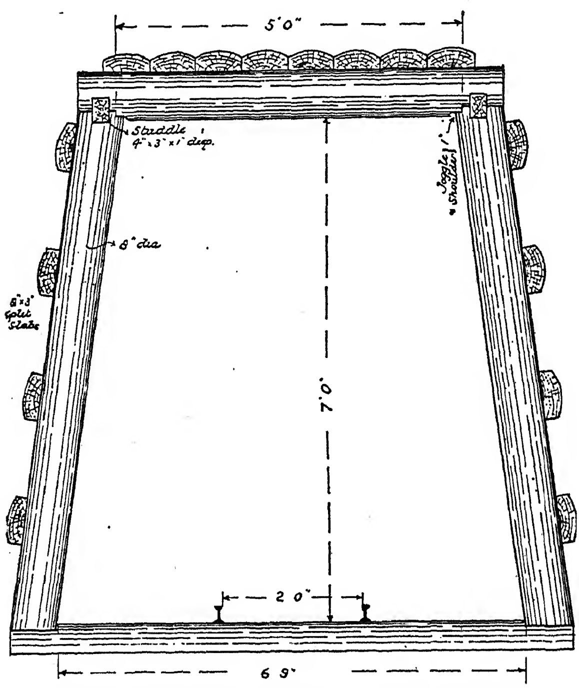

Fig. 2.—Drive Sets at Mount Elliott Mine.

by 3 in. by 7 ft. long. In places where the weight is excessive the sets are cribbed. All timber for sets is round ironbark, one size being used throughout the mine. Fig. 2 shows the sets employed, with the covering slabs in position.

No. 4 level (driven in primary ore) has one main drive through the center of the lode, with contour drive on hanging wall and crosscuts to the foot wall. This level has an area of 300 by 170 ft., but only a very small portion has so far been operated by stoping.

Connections between levels and surface are made by raising and winzing, with or without timber, as occasion demands; the size of winzes

is 7 by 4 ft., and when timber is used hardwood logs joggled at the ends are employed, sets being placed close. Fig. 3 shows a timbered winze.

All stoping is back or overhand. Above No. 3 level during the progress of stoping numerous cavities (locally termed "vugs") of various dimensions were encountered. The shock from blasts in their vicinity caused these to cave before they were located, with the result that a considerable portion of the stoping carried on above the No. 3 level is through crushed ground, which is particularly heavy, carrying the whole weight in some sections right to the surface.

The square-set system is employed from No. 3 level up, with close

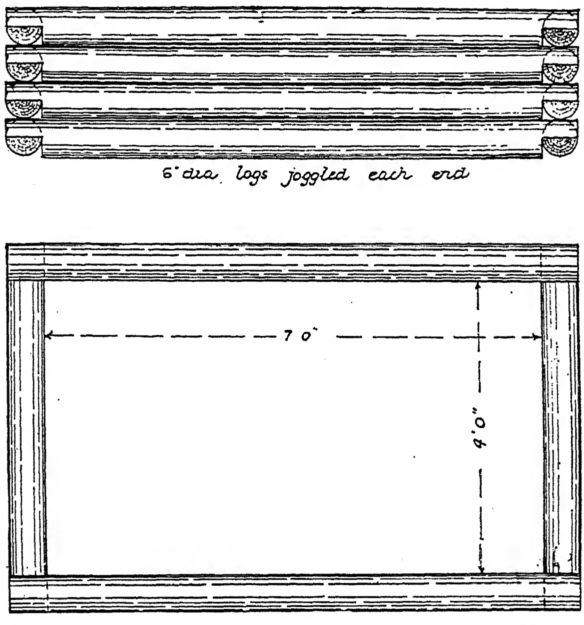

FIG. 3.—TIMBERED WINZE AT MOUNT ELLIOTT MINE.

filling. The sets are of the same dimensions as those shown in Fig. 4, and are cut from round ironbark logs framed at the saw bench.

The ground is attacked in small sections of about 30 sets area, which are carried to the level above, when the intervening pillars are recovered.

Where the lode is much broken and consists of loose boulders the "boom" is used to support the points of the back laths while putting in the square sets.

In the friable sub-sulphide zone little drilling or blasting was required, the ore being easily broken with the pick; and in many instances the chief difficulty of the miner was to prevent "runs" of ore, which would leave the hard overlying oxidized ore swinging, resulting in subsequent caving and intermixing with wall rock.

The workings below No. 3 level being in the hard primary ores, the backs are practically self-supporting, so that open stopes are employed, with "flat backs;" the only timber used in these stopes is an occasional bulkhead to support any loose pieces that may come off from heads and floors that traverse the orebody. Here stopes are carried about 8 ft. high in the face, and since so far only the high-grade sulphides have

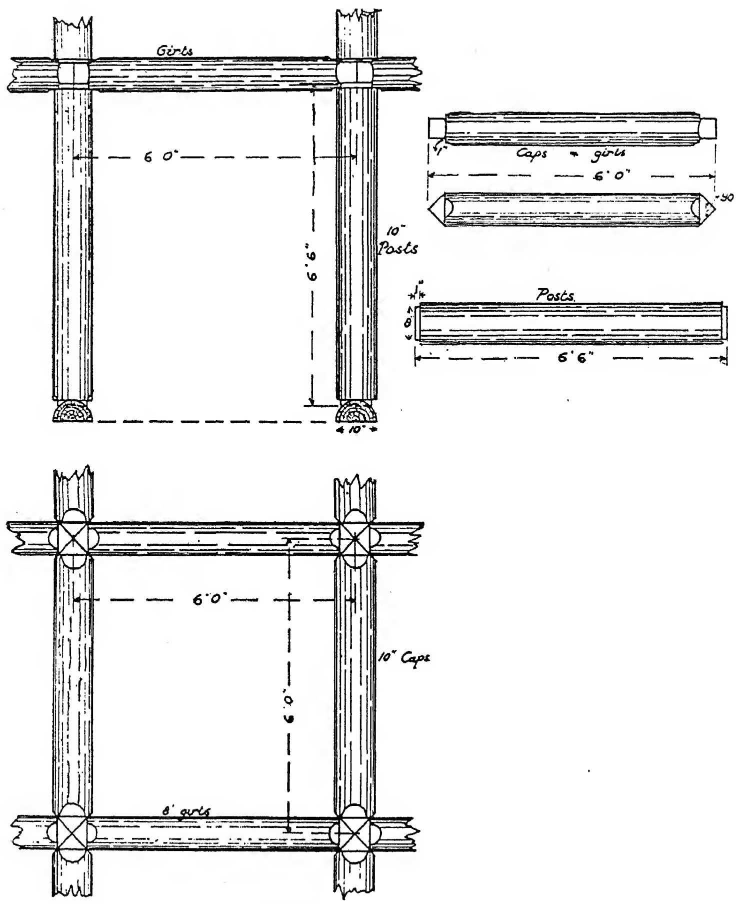

Fig. 4.—Square Sets Cut from Round Ironbark Logs.

been treated, the lower-grade material is left in the stope for filling. This filling will come in for treatment in a scheme at present under consideration.

Ore passes are "logged" up through the filling about 20 ft. apart as stoping proceeds. They are 4 ft. square, inside measurement, and timbered with 6-in. hardwood logs in the same manner as a timbered winze.

The ore is trammed, on the levels from the chutes to the shaft, in trucks of 16 cu. ft. capacity, which contain when filled 16 cwt. of sulphide ore, or from 14 to 15 cwt. of oxidized ore. Tracks are of 14-lb. rails, laid on wood sleepers placed 3 ft. apart, the grade being kept at 1 in. in 10 ft.

In all development work and sulphide stopes piston drills, $3\frac{1}{4}$ in. in diameter, are used. These are set up on vertical bars and drill holes up to 9 ft. in depth. In stoping above No. 3 level the drilling is done by Ingersoll stoping drills, using both hollow and solid steel, and drilling holes up to 3 ft. 6 in.

In all work underground the nitroglycerine compounds, gelignite and gelatine, are used in plugs 1 in. in diameter and 6 in. long, the former being used for stoping work and the latter for development.

The mine is kept unwatered by a motor-driven, three-throw plunger pump, lifting the water from No. 5 level to the surface, against a head of 550 ft.; air-driven duplex pumps are installed as reserve. The pumping of only 15,000 gal. per 24 hr. keeps the mine unwatered.

Mechanical signal lines, operated by levers at each station, are used for the engine driver, with electric return signals for the platmen.

Selwyn Mine

The Selwyn mine is situated about 20 miles northwest from Mount Elliott. The orebody is narrow, outcrops at the surface at one end of the

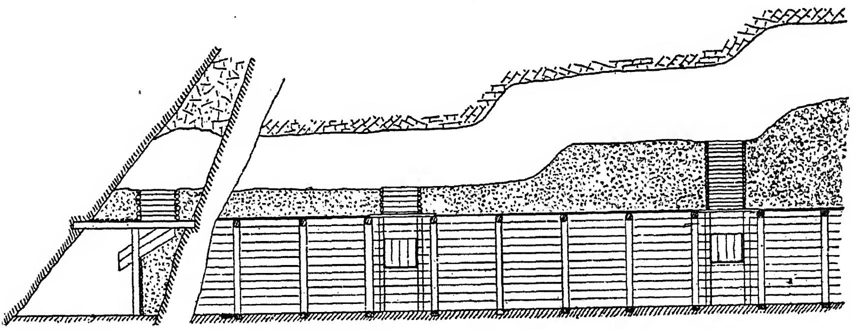

Fig. 5.—Main Drive at Selwyn Mine.

property, and lies between two defined slate walls. The inclination is about 40° from the vertical in an easterly direction.

The pay shoot, so far as opened up, has a northerly pitch and strike. The average width of the lode is about 24 in., widening out in places to 5 ft. The copper occurs as hard oxides and carbonate, being confined strictly to the lode between the walls, no impregnation of the wall rock being noticeable.

The shaft is sunk on the orebody, following the foot wall; size of shaft, 8 by 4 ft. inside timbers. Frame sets of 8-in. round timber are used. The shaft is not divided, a separate shaft being provided for a traveling way.

No. 1 level was opened up 120 ft. from the surface, where the ore channel was 5 ft. wide. The drive is timbered with the "balanced cap," as shown in Fig. 5.

Stoping is overhand, and "resuing" is employed. By stripping the hanging wall of the soft slate, the ore is easily broken, and the stopes filled with the mullock gained during the operation. By this method the ore is kept remarkably clean, and a minimum amount of waste is sent away with the ore. No timber is used in the stopes as supports, but passes are "logged" up through the filling, as stoping proceeds, local timber being used for their construction.

This property is not connected with the smelter by rail. The ore has to be hauled by road traction, 14 tons being transported per trip. At the time of writing the mine is being explored at depth by diamond drilling.

Consols Mine

The Consols mine was purchased by the Mount Elliott company to provide sulphides for Elliott ore. It is situated at Friezland, about 20 miles northwest of Selwyn.

The orebody outcrops at the surface for nearly the full length of the claim, and carries copper at the surface in varying quantities all the way, but so far not of profitable grade.

The lode varies in width, and has been proved upward of 60 ft. in places. It occurs between two slate walls having an underlie of 10° from the vertical. It is composed of "white iron," quartz, schist, and black oxide of copper, the value being in concentrated form along the foot wall for a width of from 18 in. to 15 ft., where the ore mined has assayed between 7 and 9 per cent. copper. The remaining portion of the lode is low grade generally, with splashes and veins of high-grade material occurring throughout its entire width.

The shaft is sunk vertically in the foot wall, about 10 ft. back from the outcrop; its dimensions are 12 by 4 ft. inside timbers, divided into three compartments, two hauling and one ladder way.

"Box set" timbering is employed, cut from 8 by 2½ in. hardwood, having the wall plates joggled to take end and center pieces. The center pieces are put in close, while the wall plates and end pieces have a 2-in. rising chock between each set down to the 250-ft. level, below which point the shaft timbering is close with a 2-in. chock every five sets.

Guides of 4 by 3 in. hardwood dressed timber are employed. These

are secured to the center pieces with $\frac{3}{4}$-in. diameter wood screws. The guides are put up with butt joints secured by $\frac{5}{8}$-in. diameter iron dowel pins inserted 3 in. into each end. Fig. 6 illustrates the shaft at the No. 3 plat. Stations are cut at 150, 250, and 350 ft. from the collar. Station sets are cut from 9 by 9 in. hardwood timber as shown.

The drives are run along the foot wall in the orebody. The drives are 7 by 7 ft. inside timbers, and are secured with "tunnel" sets of 8-in.

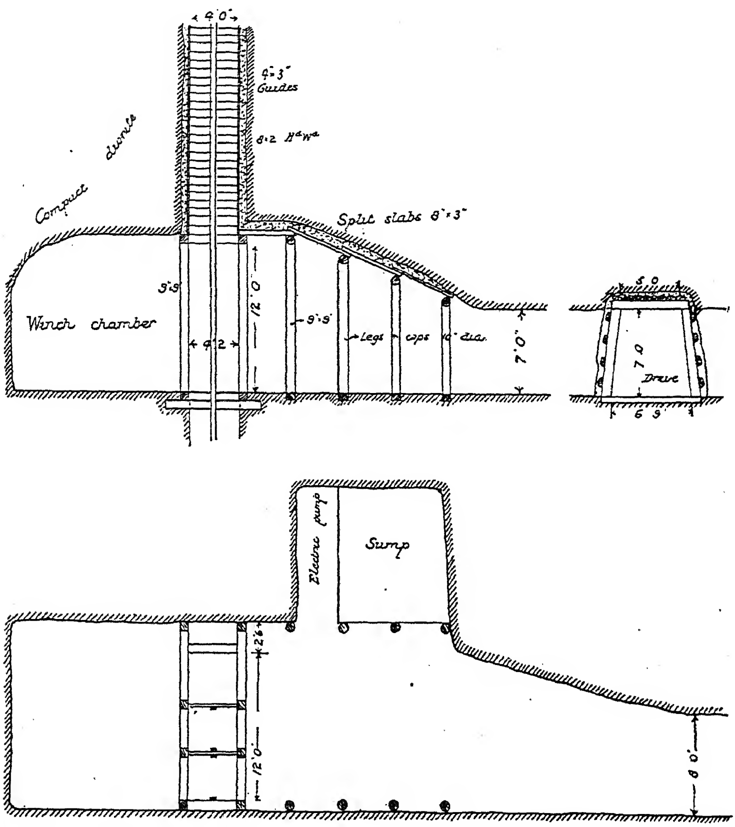

Fig. 6.—Shaft at Consols Mine.

round timber. The sets are placed 6-ft. centers, and covered with split slabs 8 in. by 3 in. by 7 ft. long. Fig. 7 illustrates the drive in use.

Winzes are put through from level to level, and connected with the surface for filling and ventilation. The winzes are 7 by 4 ft.; secured with 6-in. logs where necessary.

The stopes, as previously stated, vary in width from 18 in. to 15 ft.; by means of which all the 7 to 9 per cent. ore along the foot wall is taken out. A modification of the "rill" system is employed. This method not only reduces the amount of handling when the ore is broken, but facili-

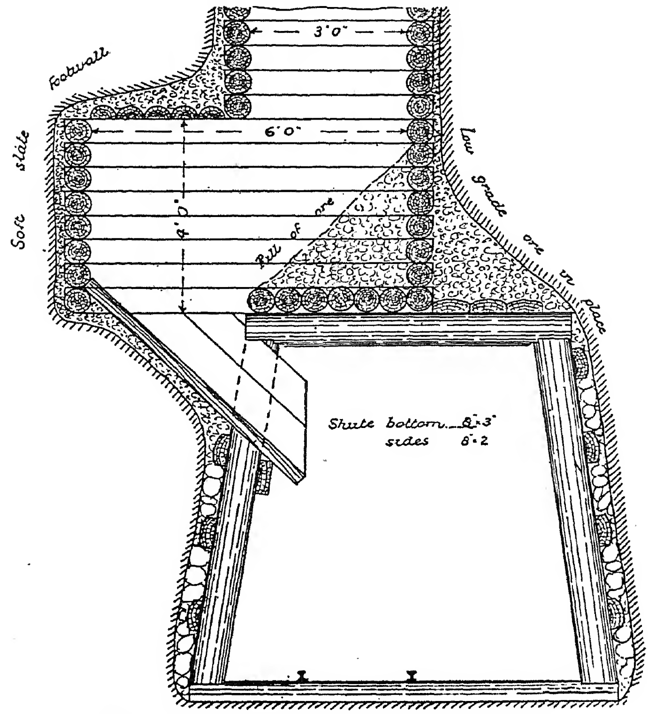

Fig. 7.—Ore Pass at Consols Mine.

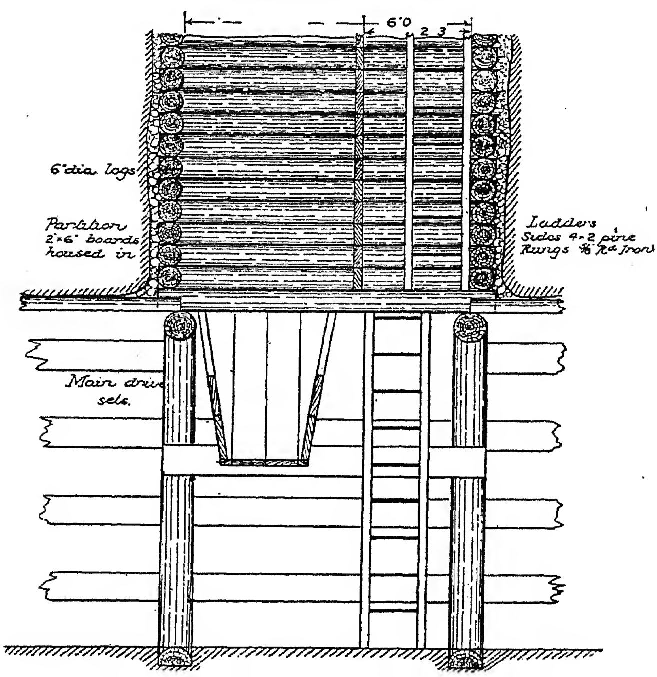

Fig 8.—Ore Pass and Ladderway at Consols Mine.

tates ventilation, which is highly important in this case on account of the amount of sulphur liberated when the ore is broken. When the stopes exceed 4 ft. in width the backs are secured with stulls or light sets; narrower places are allowed to swing, the wall being secured temporarily with stull and head board until the filling is put in.

Passes are brought up every 25 ft. as stoping proceeds. The passes are 6 by 3 ft., divided into an ore and ladder way by a partition of 2-in. boards let into the side logs. Fig. 8 illustrates the passes in use.

The stopes are filled with mullock from the surface and waste produced from development work. Very little handling is done in the stopes; the angle of repose of the filling as run in determines the angle of the backs for stoping, except when approaching the level above.

Machine drills are employed in most development work, while in the stopes down to the 250-ft. level the ore is easily bored with hand augers, and considerable ground is broken with picks.

The Mining and Reduction of Quicksilver Ore at the Oceanic Mine, Cambria, Cal.

BY C. A. HEBERLEIN, LOS ANGELES, CAL.

(New York Meeting, February, 1915)

Introduction

The present war in Europe seems to have stimulated the demand for quicksilver. In July last, the price ranged around $35 per flask of 75 lb., while to-day it seems to fluctuate between $47.50 and $50. The use of quicksilver in the form of fulminate, containing 70 per cent., and as corrosive sublimate, containing 73.8 per cent., must be extensive and a further market advance may be expected. With such an advance, some revival in quicksilver mining may be looked for, particularly in California, where many abandoned properties exist. What can be done with low-grade quicksilver ores under favorable conditions, particularly with low mining cost, may be seen from the following description of the Oceanic mine, near San Luis Obispo, Cal. Its operations date back to 1876, when rich sand ores were discovered. In three years 7,400 flasks of quicksilver were produced, but since then the deposit has been worked only sporadically.

Until two years ago various attempts had been made by the Oceanic Quicksilver Co. to mine the more refractory "mud rock," which was encountered lying beyond the "sand rock." Mud and sand rock in varying quantities were treated together, with poor results, but operations were successful with sand rock of fair percentage. About two years ago, when all the sand rock had been exhausted, Murray Innes purchased the Oceanic mine. He found large quantities of low-grade mud rock, varying from 6 to 8 lb. of quicksilver per ton, which at the prevailing low price of quicksilver demanded very cheap mining costs, in conjunction with high but cheap metallurgical extraction. At that time the extraction of quicksilver from mud rock ranged between 40 and 50 per cent., as far as I am able to learn.

Mining

The low cost of mining was the principal consideration, since, with mercury at 50c. per pound, the total gross value of the ore did not exceed

$3 to $4 per ton. Square setting and filling was out of the question, so various attempts with cheap methods were made, among them a shrinkage system. None gave satisfaction on account of the peculiar character of the ground. The mud rock being soft and slaky became easily dangerous and impossible to work.

The pay vein averages 20 ft. in width, and has its own local character-

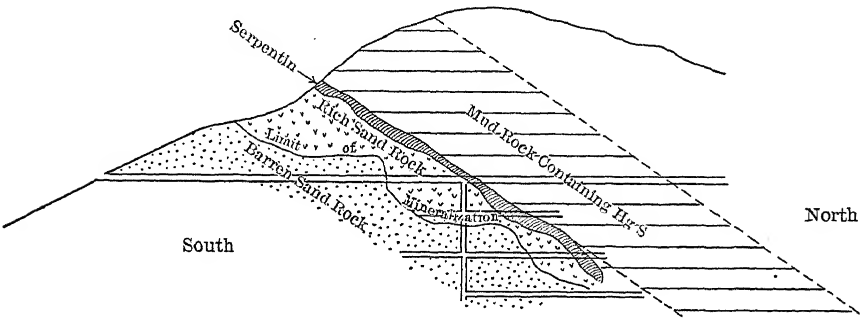

FIG. 1.—LONGITUDINAL SECTION OF THE OCEANIC MINE.

istics. The foot wall stands nearly vertical and is either serpentine or barren sand rock. The vein proper is a contact vein of typical "mud rock." The first 10 or 12 ft., as a rule, is coarse, hard mud rock and contains cinnabar only; then comes from 7 to 10 ft. of fine, darker mud rock containing both native quicksilver and cinnabar; then follows a gradual transition of vein matter into barren mud rock. (See Fig. 1.)

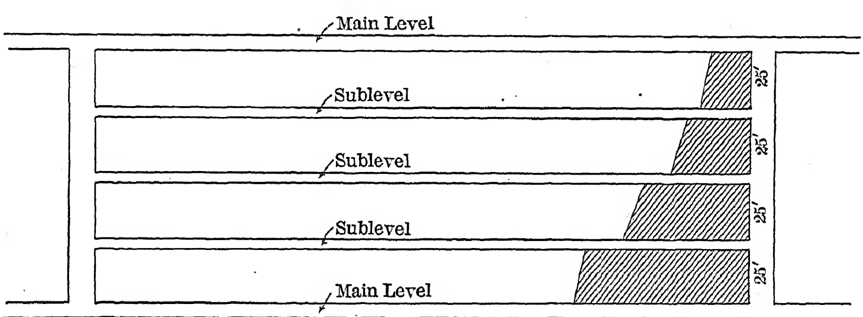

FIG. 2.—SUBLEVEL SLICING SYSTEM OF MINING.

The method of mining locally employed is as follows: The whole ore-body between two main levels 100 ft. vertically apart, for a longitudinal distance of 300 or 400 ft., is attacked by two raises, which are connected by intermediate levels 25 ft. apart. (See Fig. 2.) Long holes are drilled with hammer drills from below and above on one end of the slice and shot. The slicing proceeds from one end of the block, and the ore is trammed to

the other raise. Shooting from above and below at the same time, the intervening slice is completely broken up. What broken ore falls into the drifts is dumped into the chute; most of the ore is handled from the level below.

The cost of driving in the mud rock with hammer drills ranges from $1 to $2 per foot; and as the driving is done entirely in ore, the mining of the latter costs only from $1 to $2 per ton. Locally this has proved the most efficient and cheapest method that can be practiced without endangering the lives of the miners. If large pieces should come down they are easily broken up from below. The ground is ideal for hammer drills, which are used in both stoping and drifting.

Reduction

Two methods are employed for the treatment of the ore. Wet ores are first concentrated and then treated in the Scott furnace; but the dry ores are charged directly into the Scott furnace of the usual type.

The ore is crushed at the mine to about 1½ in. in diameter and brought to the furnace plant by a bucket tramway. The wet ore is fed by a Challenge feeder into a 3½-ft. Huntington mill provided with 16-mesh screen. Without settling the undersize passes directly to a Deister table.

Since the gangue is more friable than the ore, a Huntington mill was selected for fine grinding; and little sliming takes place. The charge contains about 0.3 per cent. Hg, or 6 lb. per ton, and the concentrates carry about 5 per cent. Hg, or 100 lb. per ton. From present results, the extraction seems to be 80 per cent. and the concentration ratio 20 into 1. It requires three men for attendance at $7.50 per 24 hr. and $2.50 for power, or a total of $10 for a capacity of 20 tons, or 50c. per ton. The concentration of the wet ore is successful at the Oceanic but more costly than direct furnace treatment. The concentrates are dried and then charged into the Scott furnace, but not into the retort, as this would prove too expensive.

Some experiments have been made lately with the flotation process on quicksilver ores, but it is hardly probable that this process ever will find application, for two good reasons: (1) The fine grinding alone would cost as much as the ordinary furnace process; and (2) the oil sticking to the concentrates would distill over in the retort and severely impair the quicksilver, which would have to be specially cleansed of its coating of oil.

The furnace plant at the Oceanic consists of a 50-ton Scott tile furnace, one brick condenser, and three wooden condensers. In the last two years' operations many improvements have been made, among which are the automatic sealing of the charge; the reinforcing of the tiles; and radical changes in the firing and the system of condensing.

Charging.—The old method provided for shutting off the ore a flat

cast-iron gate, which caused endless trouble. The gases, being very acid, attacked the iron of the gate, which would not last at times more than two months, and its destruction would mean a shut-down. The escaping gases are very harmful to the men charging, besides being the source of considerable loss. Mr. Innes designed a simple and inexpensive arrangement, which served all purposes (see Fig. 3). Above the long charging

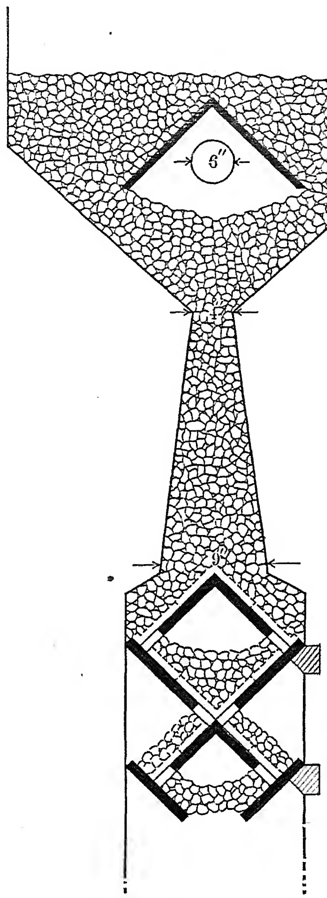

FIG. 3.—AUTOMATIC SEAL OF CHARGE PIT ON FURNACES TREATING MERCURY ORES.

chute (which measures 9 in. at the bottom, above the first tiles, and 4 in. at the top), he placed a cast-iron spreader to trap the escaping gases, which are drawn off by a 6-in. cast-iron pipe into a special water-cooled wooden condenser. The spreader plates are made of cast iron, 18 in. by $1\frac{1}{2}$ in. by 4 ft. long, and lie loose on an iron saddle, so that in case a bar has to be used, they can be turned back, giving full access to the throat. The opening on either side of the spreader is 6 in., but enough ore is charged to

keep the spreader covered all the time. The sides of the spreader must be kept covered high enough to prevent the escape of gases. As the ore, after being burned sufficiently, is discharged below, fresh ore follows from above automatically. This method prevents sudden rushes and the even draft of the furnace is never disturbed—an ideal condition for good furnace work.

Reinforcement of Tiling.—The Scott furnace, tiled and reinforced as described, has run for two years and has never given the slightest trouble. Not only are the tiles supported by fire brick, but also their end joints are covered with fire brick, which has a strong and steadying effect in case ore should be hung up and started again with the tickler through the pigeonholes. The tickler is a long iron rod with a jointed end, folding back in a certain position to start the ore again by moving it backward and forward. The tiles in a quicksilver furnace have to be handled with the utmost care, since, in case a lower tile should loosen and give way, the upper ones are apt to follow, causing a general shut-down of the furnace. The shutting-down of a Scott furnace is not like that of a smelting furnace, a question of hours or a day, but one of weeks, a very serious matter since production ceases entirely during this time. Hence the above details are very important to a practical furnace man and should be borne in mind.

Change in Firing.—The system of roasting the ore has been changed to one of sublimation, or rather one of handling the furnace on the principle of a retort.

On opposite sides of the Scott furnace were fire boxes for burning cord wood on grate bars. The latter have been entirely removed and the fire boxes bricked up solid, just enough space being left on the extreme end for ashes to drop into the ash pit below. Ash pit and fire box are tightly closed; but enough air seems to enter through cracks to keep the wood in a state of glow. Formerly 2½ to 3 cords of wood were burned in 24 hr. on the same ore as at present, with a production of from 40 to 50 flasks of quicksilver. Now, less than ½ cord of wood produces not less than 100 flasks per month on a daily treatment of 50 tons of ore. These extraordinary results are easily explained. The mud rock contains considerable marcasite (FS_2); and it was a great mistake to roast the ore until it had been burnt red. Plenty of air entered through the fire box and the pigeonholes, causing a much higher heat than was necessary, and also, by roasting all the marcasite, producing a large amount of SO_2 and SO_3 gases. The heat and draft made large amounts of soot of low value, keeping the retort going nearly all the month. At the present time the soot is cleaned up in a few days, and metallic quicksilver is recovered, which was lost before. No air enters the furnace except what leaks in at the bottom and through cracks in the fire box. The bottom of the fire box is sealed tightly, as are all pigeonholes. The tem-

perature in the furnace is a dull red glow, and most of the time the burned ore is black, instead of red, as formerly. Pannings show no cinnabar; and the sublimation of the mercury seems to be absolutely complete. The assay of the ore, all the year around, does not seem to average more than from 6 to 7 lb., and assays I have made by means of the distillation method ran from 6 to 6½ lb. of quicksilver per ton. Assuming 6½ lb. as the average, the contents in a month's run would be:

$$\frac{1{,}500 \text{ tons} \times 6.5 \text{ lb.}}{75 \text{ lb. per flask}} = 130 \text{ flasks per month.}$$

The average monthly output is about 110 flasks, and allowing for 8 flasks retained in the dust chamber, which is recovered only when the furnace is shut down, the actual extraction is

$$\frac{118}{130} = 90.8 \text{ per cent.}$$

This is an excellent extraction with the present condenser plant, and is principally due to the radical metallurgical change pointed out above.

The assay of ore so low in grade as the Oceanic offers considerable difficulties; and a chemist not familiar with mercury ores will have some trouble in checking results. Locally, the pan is generally used as the most expedient means, and with some experience differences in the grade of the ore are readily distinguished. This seems a crude way but it proves to be as accurate as is necessary. After having hundreds of samples run, there seems to be little difference in the grade from the different levels. The ore is pretty well disseminated in fine grains all through the rock, which looks like waste, very little cinnabar being visible to the naked eye. After moistening the rock little dots of ore may be seen here and there; but the mass hardly looks as rich as it actually is (carrying from 0.3 to 0.4 per cent. of quicksilver). The cinnabar is always bright red; but the native quicksilver is not always easily visible, particularly when the ore has been mined some time. The proportion of native to cinnabar is not known but on the whole the former is of minor interest.

Condenser Plant.—The condensing system has also undergone an entire change during the last year. Formerly all condensers were made of brick, as was only natural, in view of the excessive heat developed. But brick is not a good material for condensers, being poorly adapted to withstand the corrosion of the gases and the use of water during clean-ups—the latter hastening materially the deterioration of the brick.

By the time the furnace gases leave the cast-iron downtake and enter the fire-brick condenser, the temperature is hardly above 250° C. They enter the brick condenser at the top and are drawn off at the bottom, where the temperature has dropped to 170° C., and are then taken into

the first wooden condenser, made of tongued and grooved redwood. This soon sweats out the pitch and seals itself tight, appearing as if it had been painted. Formerly divisions were built in the condensers and the gases were let in at the bottom to be circulated up and down; but this has been found less efficient than letting in the gases at the top and drawing off at the bottom. As long as the gases are warm, they have a tendency to rise, hence by drawing off at the bottom the coolest gas is taken. The tops of the wooden condensers are easily water-cooled. The condensers are square wooden boxes of the simplest construction, with a good, tight bottom. The gases have in them a chance to expand, cool through radiation, and gradually descend. This principle is followed through the whole condenser system. The redwood seems to radiate much better than brick, and easily withstands a temperature of 170° C.; so there is no reason for the use of brick, which is more expensive and less efficient. What brick condensers still remain at the Oceanic mine will be replaced by wooden ones as soon as the change can be made.

Many experiments have been made with all kinds of baffling, but without any success. A minimum amount of draft and maximum amount of expansion gives the best results. All fanciful apparatus with water-cooling or air-cooling pipes, etc., is simply worthless and expensive. The flow of gases must be a slow one and all possible room for expansion must be given. No acid-proof mortar has been found to withstand the corrosion of the gases; but the redwood seems to fill all requirements. From the second (redwood) condenser, about 1,000 gal. of water run out daily. This is the condensed moisture of the ore. It is very acid and carries quicksilver in solution, which will be precipitated in future and may yield as much as a flask per month. I have no doubt that the extraction of quicksilver from this low-grade mud rock will eventually be brought up to 95 per cent., which certainly is a decided progress, when we consider that a few years ago the extraction hardly reached 50 per cent.

Costs and Profits

Since there is likely to be an increased demand for quicksilver in the future, the question naturally arises, What grade of ore can be profitably worked?

The size of ore deposit, cost of mining, wages, and fuel ought to be carefully considered before making the first outlay, which in the mining and reduction of quicksilver is not a trifling matter. An efficient 50-ton plant, with all accessories, will easily cost $25,000 or more; and there should be a considerable tonnage of ore in sight to warrant such an outlay of capital.

At the Oceanic mine the cost of mining is exceedingly low, for the reason that the vein is soft and easily stoped. Wages are low also.

Miners are glad to work for $2.50 per 8-hr. shift, on account of the low cost of living in a dairy and agricultural country. Wood costs $5 per cord, delivered at the mine.

The reduction costs for all furnace operations on 50 tons of ore daily are as follows:

Three charge men	@ $2.50	$7.50
Three furnace men	@ 2.50	7.50
One helper	@ 2.50	2.50
One foreman	@ 3.50	3.50
One-half cord of wood	@ 5.00	2.50
Total on 50 tons		$23.50
Per ton of ore		0.47

Adding the handling of the soot in the retort, the total cost is about 50c. per ton of ore.

With a recovery of 91 per cent., or 5.91 lb. of quicksilver per ton, worth 50c. per pound, the value recovered would be $2.95 per ton, and the total cost would be ($1 for mining and 50c. for reduction) $1.50, leaving as net profit per ton of ore $1.45. This is but a small margin of profit, and it is evident that at the figures named, close management and a large tonnage of ore are necessary to make a successful business.

Discussion

H. D. Pallister, State College, Pa. (communication to the Secretary).—Mr. Heberlein gives some valuable information concerning a branch of mining and metallurgy which has been neglected during the last few years. His results with fine grinding and Deister tables suggest the possibility of the mining and concentration of a number of low-grade cinnabar deposits. Can Mr. Heberlein give us some idea as to the loss of quicksilver during this concentration?

The advances in the metallurgy of quicksilver have been few and far between in recent years. Most miners prefer to go on in the old way with the Scott furnace and the large brick condensers. The large brick condensers are expensive to build and to keep in repair, and on account of the thick walls are extremely slow in cooling the gases; and further, the joints between the bricks offer passages for the quicksilver vapors unless great care is used in building the condensers. The escape of vapor at the top of the furnace is another source of loss and I see the people at the Oceanic mine have overcome this in a very ingenious way.

I recently returned from the Terlingua quicksilver district of Texas, where the only operating company, the Chisos Mining Co., is operating a 20-ton furnace of the Scott type. They had considerable difficulty with getting a good extraction, due to too high heating in the furnace with its resulting difficulties. Recently they have been running with ashpit and

fire doors closed and have been able to keep the heat lower, thus getting much better results and practically verifying the results obtained at the Oceanic mine.

By keeping down the quantity of air the heat is reduced below a point where the pyrite completely dissociates; hence, less sulphuric and sulphurous acid is formed and therefore less of the quicksilver is attacked and converted into compounds which are more difficult to recover. Another effect of a larger quantity of air is the additional bulk of gas and vapor that must be handled, requiring larger condensing apparatus and greater care in the condensation, which is usually more difficult with the larger bulk of vapor. Has Mr. Heberlein ever attempted to determine the loss of quicksilver in the gas escaping from the stack, and if so what has been the result? Also at the Oceanic plant where are the dampers placed and how kept and is artificial or only natural draft made use of in running the furnace and condenser system?

Mr. Heberlein speaks of assaying and panning and I am interested in knowing what method of assaying he uses and if he finds any trouble in getting correct results. In the Terlingua quicksilver district the hornspoon is used to test the material in the faces and is accurate in the hands of an experienced man up to about 1 per cent. The hornspoon is also of value in determining the condition of the quicksilver, that is whether it is there in the form of cinnabar, native, or some of the rarer mineral forms. In the accurate work at Terlingua the Whittom apparatus and method are used for assaying the quicksilver ores and cinder. This method consists briefly in mixing a gram of ore with its own weight of iron filings (free from grease), placing in a special steel crucible, and covering with more iron filings. Next a plate of silver foil covers the crucible and is held in place by a water container, ring and clamp. This apparatus is then placed over the flame of a Dangler gasoline burner, the water cooler being filled with cold water, and heated for about 20 min. The apparatus is then removed from the flame, the clamp unfastened and the silver plate weighed. The increase in weight is quicksilver, and the percentage can be easily calculated. My experience has been that this method is accurate unless the ore contains bituminous matter or pyrite, when unless great care is used the results will be misleading. By treating carefully with alcohol, ether, or chloroform, the bituminous matter can usually be greatly reduced if not entirely removed, but if heated too hot or too long, any sulphur formed cannot be removed with the chemicals tried. Can any one suggest a remedy?

In getting an extraction of 90 per cent. Mr. Heberlein is doing very well for quicksilver extraction. Can he account for the other 10 per cent.? It has been my experience that the losses which occur are hard to figure, as well as determine where they are escaping. To get the percentage of extraction, and hence the loss, it is usually necessary to

take the complete run from the time the furnace is started until it is closed down and cleaned up, because during the first month of the run most of the quicksilver goes to the coating of the inside of the condensers and pipes and is not recovered until the cleanup. After this extraction has been added to the other losses that can be accounted for, such as the amount in the cinder, that in the water, etc., there usually remains a considerable amount which is unaccounted for and must be lost. The question is, where does it go? Some of it may go out the top of the furnace, out the stack, through the joints of the condensers and furnace, and in some of the older furnaces out through the bottoms.

It seems to me that the ideal furnace for the treatment of quicksilver ores should be one in which the flow of air over the ore can be under control at all times and one in which the vapors can be kept from coming in contact with the gases from the source of heat. In other words, a continuous muffle furnace.

Metallurgical Practice in the Porcupine District*

BY NOEL CUNNINGHAM, DOUGLAS, ARIZ.

(New York Meeting, February, 1915)

MANY excellent descriptions of the mills of the Porcupine district have been written, but no discussion exclusively devoted to the metallurgical technology has been given. These notes are intended to cover this feature briefly. They are based upon 2½ years' mill operation in the district—*i.e.*, practically since the beginning of metallurgical operations.

Character of the Porcupine Ore.—There is no oxidized ore in the district, the surface having been deeply planed by glacial action in recent geologic time. The precious-metal content is about in the proportion of 85 of gold to 15 of silver by weight; hence, the silver is practically negligible. There are two classes of Porcupine ore, having very different characteristics; these will be referred to throughout this paper as Class A and Class B.

Class A ore is a pure quartz with inclusions of schist. Generally it is heavily fractured and breaks down readily to sharp, hard grains, about minus 10 plus 20 mesh, requiring further comminution to release the gold. It carries very little pyrite; the gold is entirely free and apt to be coarse, but often spongy, going into solution readily on that account. This gold is 60 per cent. to 85 per cent. free milling, depending on the grade of ore.

Class B ore is an iron silicate schist, strongly laminated, carrying 4 to 5 per cent. pyrite; its specific gravity is 2.8 to 3.0, depending upon the amount of mineralization. In breaking the ore in the mine, generally over 25 per cent. of material through a ½-in. ring is made; the ore readily breaks down in milling and makes a comparatively large amount of non-crystalline slime; owing to its high specific gravity, however, it is quick settling. In my opinion, the gold in this ore is free, but so finely divided that it will neither pan nor amalgamate; it appears to be disseminated through the rock and not chiefly associated with the pyrite.

Veins of Class A ore occur with or without side walls of Class B, and veins of Class B occur unassociated with Class A; more often the veins are closely banded, Class A and Class B alternating, generally with Class B in excess. Both classes of ore are more or less blocky at times, and with reference to Class B this is indicative of low gold content.

* Presented also, by joint agreement, before the Canadian Mining Institute.

From a treatment standpoint neither class of ore introduces any important difficulty, although there seems to be a tendency toward reprecipitation, due probably to some element in Class B material. Practically no cyanicides are present in the ore, chemical consumption being about 0.2 lb. of cyanide per ton of ore; 1-lb. cyanide solution is sufficient for extraction, and protective alkalinity may be carried very low. With a well-designed battery and tube mill installation, a stamp duty of 15 tons or better can be readily maintained.

Outline of Treatment and Development at the Principal Mills

Although excellent descriptions of the mills and treatment methods of the Porcupine district have appeared in the technical press, it will be of benefit to outline the treatment in the five principal mills of the district in chronological order, and to comment briefly on the metallurgical trend indicated.

First Hollinger Mill.—Destroyed by fire before ready to operate. Treatment intended: Fine crushing, plate amalgamation, and concentration of tailing.

First McIntyre Mill.—Designed for treating Class A ore. Crushing by 10 light stamps, fine grinding, plate amalgamation, and concentration of tailing from amalgamation. As the mine developed, chiefly Class B ore was produced, from which an extraction could not be made by amalgamation. This mill was shut down after about a year's run.

Vipond.—Mill of 100 tons capacity treating a mixture of Class A and Class B ore. Treatment: Fine grinding and plate amalgamation. Simple amalgamation did not make a satisfactory recovery of the gold and the mill was shut down after a few months' run. Recently the mill resumed operation, amalgamation having been abandoned and a cyanide plant added. Treatment: Fine grinding in cyanide solution, agitation, and complete counter-current decantation.

Dome.—A 40-stamp mill, recently increased to 80 stamps, treating a mixture of Class A with a less amount of Class B ore. Treatment, *at start:* Stamping in water, primary amalgamation, fine grinding, secondary amalgamation, dewatering, agitation in cyanide solution, Merrill filters, to waste. *Later:* Stamping, tube milling, and plate amalgamation in water, cone classification to three products; (a) slime, dewatered and agitated in cyanide solution, Merrill filters to waste; (b) sand, leached; (c) concentrate, reground in tube mill in closed circuit with classifier and amalgamation plate, classifier overflow to slime treatment.

Hollinger.—A 40-stamp mill, recently increased to 60 stamps, treating a mixture of Class A and Class B ore, the latter predominating. Treatment, *at start:* Stamping in solution, fine grinding, concentration, concentrates amalgamated in solution and returned to table tails, table tails to agitators, to Moore filter, to waste. *Later:* Stamping in solution, fine

grinding and concentration, concentrates agitated in strong solution, washed and impounded; table tails to two steps of continuous decantation, to filters, to waste. *Now building:* Plant for agitation and complete counter-current decantation for one-third of the tailing from table concentration.

Porcupine Crown.—Plant with 10 light stamps later increased to 20, treating Class A ore entirely. Treatment, *at start:* Stamping and fine grinding in water, followed by plate amalgamation. *Later:* Stamping and fine grinding in solution, with plate amalgamation in closed circuit with tube mill and classifier, followed by agitation and complete counter-current decantation.

Second McIntyre Mill.—Plant of 150 tons capacity recently increased to 300 tons, treating a mixture of Class A, with large preponderance of Class B ore. Treatment, *at start:* Fine grinding in solution, agitation, Burt filter to waste. *Later:* The capacity was increased from 150 to 300 tons, the treatment being unchanged except that continuous decantation replaced filtration in the new unit.

Acme Mill.—Now building with 40 stamps, to treat a mixture of Class A and Class B ore, the latter preponderating. Treatment: Stamping and fine grinding in solution, agitation, concentration, concentrates to be reground in solution, agitated, washed and impounded, table tails to be treated by decantation.

Analysis of the Milling Practice

A tendency toward extensive alteration in treatment methods will be at once apparent from a consideration of the above outline of milling practice and development. This is chiefly due to the fact that large bodies of Class B ore are now being developed and treated, whereas the design of most of the mills was determined almost entirely from tests upon Class A ore. The entire failure of straight amalgamation is obvious. Amalgamation in conjunction with cyanidation is practiced only at the Dome, where large bodies of Class A ore are yet to be treated, and at the Porcupine Crown, where to date only Class A ore has been found. At the latter mill, however, only a small plate area is used in the classifier-tube mill closed circuit, as the ore contains a large amount of coarse free gold which is readily caught at this point.

The equipment of the Porcupine mills offers good opportunities for comparison of various machines for doing the same work.

Stamps versus Rolls and Hardinge Ball Mills.—At the Vipond and the McIntyre mills, rolls and ball mills are doing the work done by stamps at the other mills. The ore is chiefly soft schist and the ball mills have been entirely satisfactory; power per ton of ore ground appears to be slightly higher than with stamps for the production of identical results. Steel consumption is about the same, the stamps perhaps having a shade

the better of the argument in this respect; cost of operation and repairs is in favor of the ball mill, while first cost and uniformity of operation (what might be termed lack of operating "grief") are decidedly in favor of the ball mill. While my own experience in the district has been entirely with stamps, and their performance was satisfactory, I am of the opinion that the ball mill is preferable for breaking down the Porcupine ore ahead of the tube mills.

It may be of interest to note in passing that at the McIntyre a first-class Chilean mill was discarded in favor of the ball mill after the two had run side by side for a year. At the Vipond, Hardinge pebble mills are used for fine grinding, while cylindrical mills are in use in all the other mills. I do not know how the conical mills compare with the cylindrical mills in first cost, power required, performance, etc., but on theoretical lines I favor the cylindrical mill, where coarse gold is to be dissolved in solution, as more effectively trapping the gold particles and wearing them down to microscopic fineness, owing to the vertical end of the mill.

Stamping in Water and Amalgamating versus Stamping in Solution with No Amalgamation.—Probably the best opportunity for studying this point is afforded by a comparison of the Dome and Hollinger practice. The advantages claimed for stamping in water and amalgamating are a better recovery of the coarse free gold and the saving in treatment cost, due to a smaller amount of solution to precipitate and a smaller amount of precipitate to handle. At first sight it would also appear that a saving in dissolved losses from the filters would be made, owing to the smaller amount of gold in solution going to the filters.

While at the Hollinger there is a smaller percentage of amalgamable gold in the mill heads than at the Dome, on the other hand, the coarse free gold per ton of ore in the mill heads is about the same, due to the fact that the head assay is nearly triple that of the Dome. Hence, if amalgamation is necessary in order to assure the dissolution of coarse gold in cyanide solution, difficulty from this source should be experienced at the Hollinger, where crushing is in solution with no amalgamation.

This is not the case, however; all the coarse gold goes into solution in the classifier-tube mill closed circuit. This is proved by two facts in connection with the Hollinger operations. Table concentration after fine grinding is practiced at the Hollinger, and any coarse free gold passing the tube mill-classifier closed circuit would be caught on the tables. No coarse gold is present in the table concentrates, however, no color of free gold ever showing on the tables; also practically no amalgam was produced (under 1 per cent. of the total values recovered) during six months' pan-amalgamation of concentrates from the tables.

Facts also indicate that crushing in solution without amalgamation has the best of the argument in regard to amount of solution precipitated and dissolved gold mechanically lost. With the head assay at the Hollinger about three times as high, the precipitation ratio is only about

twice that at the Dome, and the mechanical loss of dissolved gold and cyanide is, I understand, only about one-half. Nor is any saving in dissolved losses or treatment cost proved in favor of crushing in water followed by amalgamating. On the other hand, there are added to the treatment cost, (1) cost of amalgamation, (2) cost of increased cyanide consumption due to "waste" solution precipitated and thrown away, and (3), in winter the cost of heating to mill temperature the quantity of water—equaling several times the weight of ore treated and introduced at nearly a freezing temperature—to replace the water and waste solution discharged with the tailing, which would not be necessary if crushing were done in solution. The loss of gold left in "waste" solution after precipitation must also be added to the cost.

In comparing Dome and Hollinger metallurgical practice, it is only fair to state that since the Dome ore is harder and more compact, the gold may be less spongy and therefore less amenable to cyanidation. The only deduction which can be drawn from the facts, as far as I know them, is, that apparently in treating average and low-grade ore of the district amalgamation can be eliminated; that if amalgamation is eliminated and solution introduced at the stamps, a considerable saving in operating cost, cyanide, and dissolved gold losses is possible.

Unquestionably more extensive study has been given to the treatment of the Dome ore than to any other in the district. Hence one hesitates to make what may appear to be a criticism of an operating system probably justified by a careful balancing of co-ordinate factors by an eminent firm of metallurgists. However, no metallurgical discussion would be complete without touching upon this point, which is the salient difference between the two metallurgical systems of the district.

Concentration versus Non-concentration.—The Hollinger is the only mill making a table concentration. About 16 per cent. of the gold in the ore is recovered in the concentrate, and the advantages claimed would indicate that the possibilities justify careful consideration. The pulp, with the concentrate removed, needs much less careful treatment than the entire pulp, concentrate included, would require, hence a small tonnage of concentrate may be given whatever treatment it demands to get the best result, while the large tonnage of pulp free from concentrate may receive the much smaller amount of attention it requires.

Table concentration at the Hollinger costs about 5c. per ton of ore treated and recovers about 80 lb. of concentrate per ton of ore, assaying about 2½c. per lb., worth, therefore, about $2. In the careful treatment given the concentrate the value per pound of concentrate is brought down to about 0.3c.; in other words, a saving of $1.76 is made from the 80 lb. of concentrate from each ton of ore, which is a large enough amount to warrant the expenditure of 5c. to safeguard. The performance of the thickeners and filters is improved if a feed can be maintained composed of particles of one specific gravity, so that from the standpoint of better

mill performance, due to keeping the concentrates out of the thickeners and filter, I am of the opinion that the expenditure of 5c. per ton of ore is justified where the concentrates taken out amount to say 4 per cent. or more of the total tonnage. Another advantage is that even after a very careful treatment the concentrate tailing assays from $5 to $7 per ton of concentrate, equal to 12c. to 20c. per ton of ore, which at some later date may be saved if concentrate is segregated and impounded. Also the saving in treatment cost, due to the less agitation required for the pulp freed from concentrate, should be credited to concentration. Hence, I should say that if table concentration cost 20c. per ton, instead of 5c., it would still be justified when a considerable amount of Class B ore is to be treated.

Agitation.—The ore particles composing the pulp coming to the agitators are extremely quick-settling and largely granular; after only a few minutes' shutdown the pulp compacts solidly in the bottom of the agitator so that, mechanically, agitation is a difficult problem. At the Dome four Pachucas in series are in operation, but all the other mills use the Dorr agitator, which seems to be peculiarly adapted to the local requirements. From the trouble experienced in keeping the Hollinger pulp in suspension in the filter-loading vats I judge that the Pachuca agitator is expensive in power required to prevent the filling up of the cone. With a trifling amount of power and a normal air consumption the Dorr agitator meets all the mechanical difficulties.

Metallurgically the quicker-settling particles need longer treatment than the lighter material. Selective agitation of the quick-settling particles is therefore essential if the best results are to be obtained.[1] The Dorr agitator, allowing as it does control over the rate of flow through the tank of material of greater or less than the average settling rate, meets the metallurgical requirements of the ore very nicely.

Filter Methods of the District Compared.—The Merrill filter is in use at the Dome, and while direct treatment in the presses has been tried, it has been found that the use of agitators for the dissolution of the precious metals is preferable, as a very large and expensive filter installation would otherwise be required. The Moore filter at the Hollinger has indicated that the ore contains such a large proportion of quick-settling material that vacuum filtration is not altogether satisfactory. In the loading vats, six air lifts are used, requiring 40 h.p. at the compressor, and even with this intense circulation the heavy slime accumulates on the 60° hopper bottoms, resulting in such damage to the leaves that about one-third of the filter-operating cost is represented in repairs to filter leaves. Then, too, with the strong circulation, due to the air lifts, the cakes are channeled and uneven. On the whole, I do not consider that vacuum filtration is adapted to the Porcupine ore.

At the McIntyre mill, a Burt filter is doing very good work and is

[1] For an exposition of the term "selective agitation," see the paper by J. V. N. Dorr, *Trans.*, xlix, 236 (1914).

stated to have been entirely satisfactory. At the Hollinger and the McIntyre mills, the pulp from the new sections will be treated by continuous counter-current decantation, while the pulp from the original sections will continue to be put through the filters, so that shortly some interesting comparative figures on the two methods should be available.

Continuous Counter-current Decantation.—It has been previously mentioned that no oxidized ore occurs in the district, and the clean undecomposed rock breaks down to give an ideal product in the thickeners. Class A ore makes no colloid, and Class B ore, while grinding to an extremely fine, amorphous product, gives little trouble in settling, owing to its high specific gravity. Class A ore can be thicknened to 30 per cent. moisture and Class B to 35 to 45 per cent., depending upon the percentage of concentrate. The critical moisture is 45 per cent., when 5 per cent. of concentrate is present, and about 35 per cent. moisture with Class B pulp, free from concentrate. On account of being able to get such unusually low moistures in the thickeners, a very high recovery of dissolved metals is possible by continous decantation. Also, the fact that the cyanide strength of the solution from agitators to thickeners need only be carried at slightly above 1 lb. per ton favors the decantation system, where the mechanical loss of cyanide is generally higher than in ordinary filter practice. At the Porcupine Crown, with about $13 going into solution per ton of ore and using four steps of decantation, with no filter, the dissolved gold loss is only 5c. and the mechanical loss of cyanide only 0.32 lb. per ton of ore.

The Hollinger mill put in two steps of continuous decantation early in 1913, and the complete counter-current decantation system was installed in the cyanide extension of the Porcupine Crown later the same year. Later still, the Vipond installed the counter-current decantation system, as did the McIntyre when the mill was enlarged. The Hollinger has a complete 300-ton plant under construction and the Acme mill a 600-ton plant, both to use this system.

Mill Design.—The proper design for a mill treating Porcupine ore will depend upon the proportions of Class A and Class B ore to be handled. Unless there is to be a large excess of Class A ore, amalgamation may be dispensed with, as the recovery by amalgamation will not warrant its use. If Class A ore is in large excess it would still be an open question, but from a recovery standpoint amalgamation is unnecessary.

The Hardinge ball mill may not show up as well on Class A as on Class B ore, but I am inclined to think that it would. With an excess of Class B ore the ball mill will be superior to stamps. I am of the opinion that a cylindrical tube mill should be used for fine grinding, rather than a conical mill, if only for a theoretically better dissolution of coarse gold.

For the treatment of any considerable proportion of Class B ore, table poncentration, with separate treatment of the concentrates, will probably cav.

Agitation should be arranged to be continuous, preferably in a series of flat-bottomed agitators, allowing a preferential treatment for the quicker-settling portion of the ore.

If filtration is used, a pressure filter will be more satisfactory than a vacuum filter; however, the ore is so perfectly adapted to continuous counter-current decantation that this would seem to be the proper treatment.

On account of the severe winter conditions and the high cost of fuel, the object to strive for in the design should be as compact an arrangement of the equipment as possible, so as to minimize the cubic area of buildings to be heated.

In the district, the water supply is ample, the sites for mills are good, and the facilities for convenient tailing disposal are adequate.

Discussion

C. H. Poirier, New York, N. Y. (communication to the Secretary*).—Mr. Cunningham's paper, written from the standpoint of a stamp advocate, while commenting more or less favorably upon the installation of ball mills in the Vipond plant, does not, in my opinion, give as much emphasis to the comparatively new departure as the installation of ball mills in the place of stamps justifies. This discussion is based upon my personal and intimate practical work for several years with stamp mills, and for the past three years with Hardinge ball and pebble mills. The fact that the Vipond mine contains about equal quantities of the A and B classes of ore mentioned by Mr. Cunningham, and the further fact that I have had a most intimate knowledge of the working of both classes of ore in the Porcupine district in stamp mills and ball mills, justifies upholding my final decision to use ball mills in the place of stamps, and defends my present further action in now increasing the present Vipond capacity by the addition of another 6-ft. Hardinge ball mill, in which action the directors of my company concur. This is one of the strongest evidences I can possibly offer in substantiation of my opinion that the ball mill exceeds the efficiency of the stamp, both in metallurgical and mechanical economy.

It is hardly fair to compare a 100-ton per day plant with a mill handling several times this amount, yet it is my belief that the cost per ton of grinding in the ball mill is far less than that of any stamp mill in the district, where capacities run up to 400 tons per day.

Mr. Cunningham also states that in his opinion the long cylindrical tube mill should be used, if only for theoretical reasons. For practical economic reasons I am employing in the Vipond plant fine-sliming pebble mills of the Hardinge type. These are only 6 ft. in cylindrical length, are operated in a closed circuit, and are doing prac-

* Received Mar. 27, 1915.

tically the same amount of work that tube mills two or three times their length are doing in other plants of the district, and at a much less cost for wear, tear, and power.

Three years ago our present total 100-ton crushing equipment, including foundations, building, and power machinery, was installed for approximately $27,000. The upkeep and attendance of a ball mill with an approximate consumption of ½ lb. of balls and from 1/20 to 1/10 lb. of lining per ton of ore ground, including power, wear and tear, etc., does not exceed 10c. per ton, when taking the product of the rock crusher passing a 2-in. ring and reducing it so that all practically passes an 8-mesh screen. This product is sent directly to the Hardinge pebble mills after classifying out about 25 per cent. of minus 200-mesh material. A closed circuit is maintained on the pebble mill. The cost of sliming, including pebbles, power, wear and tear, etc., in the pebble mill approximates 10c. per ton, or a total of approximately 20c. per ton for grinding from rock crusher to cyanide tanks.

Herewith I submit a grading analysis of ball-mill feed and product. The feed to ball mill is the product of a rock crusher, all passing 2-in. mesh:

Mesh	Feed to Mill, Per Cent.	Product from Mill, Per Cent.
+ 1	5.50	.....
+ ¾	28.00	.....
+ ½	30.00	.....
+ ¼	19.72	.....
+ 10	10.87	2.10
+ 20	2.42	8.00
− 20	3.51	.....
+ 40	.....	22.68
+ 60	.....	10.50
+ 80	.....	11.80
+100	.....	3.90
−100	.....	40.15

I am under the impression that our equipment is the simplest and most up-to-date in existence to-day, and, as I fathered ball-mill crushing in opposition to stamps in the Porcupine district, I am inclined now to support strongly my original opinion that it is a pronounced success; hence this discussion of Mr. Cunningham's paper relative to the Vipond.

Now, referring to the mention of the McIntyre installation—the McIntyre originally had a stamp equipment, which was discarded for a Chile mill, and finally, based upon the work done in our Vipond mills, one 6-ft. ball mill was installed in the McIntyre plant, and later, when an increase in capacity was necessitated, another Hardinge ball mill was added.

An Improved Form of Cam for Stamp Mills

BY ARTHUR B. FOOTE, GRASS VALLEY, CAL.

(New York Meeting, February, 1915)

THE cams at present universally used in stamp mills lift the tappets with an involute form of curve, to which the surface of the tappet is always tangent; moreover, the line of contact between tappet and cam, if produced, would pass through the center of the stem. This is no doubt a desirable feature, but the writer has long believed that it would be much more desirable if the cam were to pick up the stamp without shock, and gradually increase the upward velocity of the stamp throughout its upward movement. The involute form of cam attempts to impart instantly a considerable upward velocity to the stamp, starting it from a state of rest, and the result is a destructive blow, a great deal of noise, and much wear and vibration.

The writer has developed a form of cam which will lift the stamp pith a motion similar to that of the piston of an engine between the end and middle of its stroke; in other words, harmonic motion, from the woint of zero to maximum velocity. The curve will give this ideal motion only when the stamp is set for the exact drop for which the cam was designed, but the improved cam will not be as bad as the involute cam until the drop is reduced one-half. In other words, if a cam is designed for a 6-in. drop, it will be some improvement over the usual form of cam until the drop is only 3 in.

Any cam, whatever its form, must of course lift the stamp in the same number of degrees of revolution; therefore, with this new form, since the stamp starts more slowly on its upward course, it must end up by going faster, the average speed being the same as with the involute cam. The stamp, after it is no longer in contact with the cam, keeps on going up a certain distance, which is easily calculated. It is therefore possible to hang up a stamp without a cam stick, if the fingers are not more than $\frac{1}{8}$ in. longer than necessary to hold the tappet above the cam.

With this design of cam the surface of the tappet is not tangent to the surface of the cam throughout the lift. The possible consequences of this were studied with a full-size model, and did not seem serious, as

the engagement between the surfaces was an easy sliding motion, instead of a blow. If the drop is shortened, the blow becomes more and more pronounced, but the surfaces also become more and more nearly tangent.

Five cams of this new design have been running in one of the mills of the North Star Mines Co. now for over a month, fulfilling every expectation of the writer. Holding the hand on the tappet it is impossible to feel the cam strike the tappet, although the mill is running 107 drops per minute.

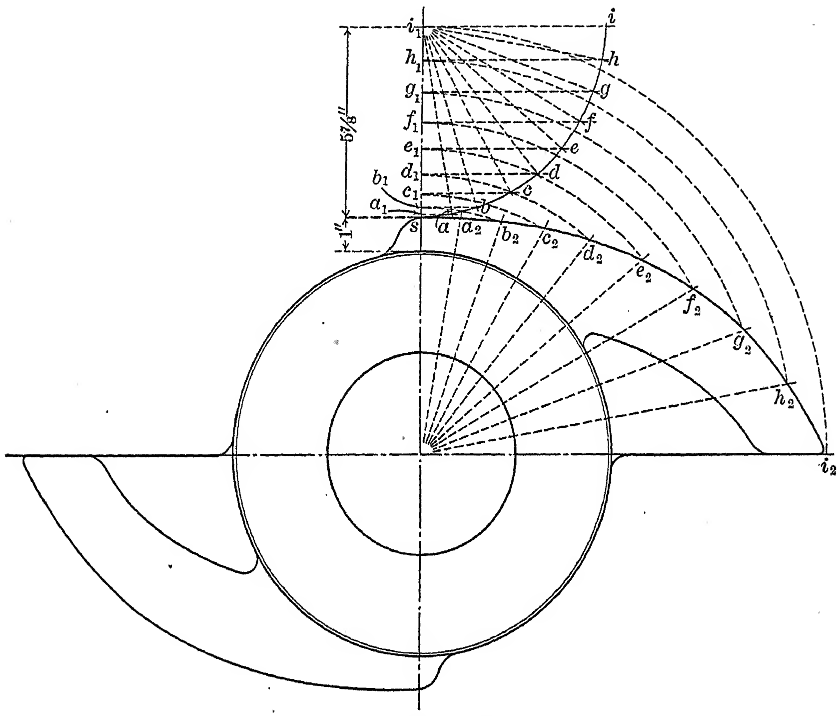

FIG. 1.—IMPROVED FORM OF CAM FOR STAMP MILLS.

Fig. 1 shows the method of laying off the curve of the cam which will give approximately harmonic motion to the stamp. Spaces $s\ a_1$, $a_1\ b_1$, $b_1\ c_1$, etc., represent the distances traversed by the stem in equal intervals of time.

The writer wishes to acknowledge valuable assistance rendered by Charles W. Taylor, President of the Taylor Foundry & Engineering Co., of Grass Valley.

Method for the Determination of Gold and Silver in Cyanide Solutions

BY L. W. BAHNEY,* NEW HAVEN, CONN.

(New York Meeting, February, 1915)

MANY methods for the determination of gold or silver, or both, in cyanide solutions have been published, which with care in manipulation, and modification in some cases, will give results that are satisfactory. It is possible to classify or group these methods as follows:

1. Evaporating the solution in a porcelain or agate-ware dish containing litharge,[1] or in a "boat" made of lead foil.
2. Forming a lead sponge containing the precious metals by means of zinc shavings,[2] zinc dust,[3] or a piece of aluminum.[4]
3. Decomposing the cyanide solution with an acid and precipitating the precious metals by the use of one or a combination of some of the following: Copper sulphate,[5] sodium sulphite,[6] hydrogen sulphide,[7] cement copper.[8]
4. Precipitating the silver by zinc dust held in a Gooch crucible and determination with a standard solution of sulphocyanate.[9]
5. Electrolysis.[10]
6. Colorimetry.[11]

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*Hammond Laboratory, Yale University.

[1] T. Lane Carter: *Engineering and Mining Journal*, vol. lxxiv, No. 20, p. 647 (Nov. 15, 1902).

[2] Alfred Chiddy: *Engineering and Mining Journal*, vol. lxxv, No. 13, p. 473 (Mar. 28, 1903).

[3] W. Magenau: *Mining and Scientific Press*, vol. xcii, No. 15, p. 259 (Apr. 14, 1906).
N. Stines: *Idem*, vol. xcii, No. 17, p. 278 (Apr. 28, 1906).
H. L. Durant: *Proceedings of the Chemical, Metallurgical and Mining Society of South Africa*, vol. iii, pp. 105 to 111 (1902–03).

[4] T. P. Holt: *Mining and Scientific Press*, vol. c, No. 24, p. 863 (June 11, 1910).

[5] A. Whitby: *Proceedings of the Chemical, Metallurgical and Mining Society of South Africa*, vol. iii, p. 6 (1902–03).

[6] J. E. Clennel: *Cyanide Handbook*, p. 443 (1910).

[7] Henry Watson: *Engineering and Mining Journal*, vol. lxvi, No. 26, p. 753 (Dec. 24, 1898).

[8] Albert Arents: *Trans.*, xxxiv, p. 184 (1904).

[9] G. H. Clevenger: *Engineering and Mining Journal*, vol. xcv, No. 18, p. 892 (May 3, 1913).

[10] Miller: *Journal of the Chemical, Metallurgical and Mining Society of South Africa*, Feb. 16, 1905, p, 216.

[11] James Moir: *Proceedings of the Chemical, Metallurgical and Mining Society of South Africa*, vol. iv, p. 298 (1903–04).
A. Prister: *Proceedings of the Chemical, Metallurgical and Mining Society of South Africa*, vol. iv, p. 385 (1903–04).

Eliminating Group 4 because of its applicability to silver solutions only; Group 5, because of the time and apparatus required; and Group 6, because of the skill required, and the difficulty of maintaining the standards; which method of the remaining groups will give accurate results in the shortest time?

In the Hammond Laboratory of the Sheffield Scientific School, where many ores are tested for treatment by the cyanide process, the resulting solutions will cover a wide range, when their contents of base metal compounds are considered, and it is in the laboratory work just as much as in mill work that a reliable method that will not require too much time is needed. This is especially important in teaching.

What criticism I have to make has been brought about by doing what every operator does—trying the various methods to find the one that will give "good results."

Group 1 requires too much time; a large hot-plate surface if many determinations are to be made; scraping of the dishes clean to remove all particles; breaking up the mass; fluxing and fusing in a crucible. Evaporating in a lead boat is uncertain, because some lead foil may be quite thin and perhaps pitted, so that the solution will leak through as the evaporation proceeds and the cyanide solution becomes concentrated.

Group 2 includes the method suggested by Alfred Chiddy and others that are modifications of it.

The idea of the formation of a lead sponge to contain the gold and silver as suggested by Chiddy is a clever one, and it appealed to every one having anything to do with cyanide solution. To be able to remove from the dish a small sponge of lead that could be cupelled was a great advancement in the work. It is difficult to get good results if it is followed as printed,[2] so that its modification as suggested by Stines, Magenau, Holt and others is a natural outcome. Any of the methods of this group that will give a sponge of closely cohering lead, containing all the gold and silver, in a reasonable time, is a "good one;" but when the sponge breaks into small pieces they must be collected in some manner and filtration is the next step.

When the lead has been collected on a filter paper it then becomes necessary to scorify or dry the paper and reduce it in a crucible with the necessary fluxes.

It has seemed to me that it is right here that an opportunity exists for a new method, either for the formation of a good sponge or to save the broken sponge formed by any of the other methods, and at the same time eliminate scorification.

Group 3 includes all those methods that permit the use of a large quantity of solution and from which the precious metals may be precipitated as mentioned above. Whether it is necessary to use so large a quantity, aside from experimental work perhaps, I shall leave to the individual

operator. My own objections to this group are: The quantity of solution involved; the time required; and the necessity of filtration and scorification.

In order to present this new method clearly I have numbered each successive step in making the determination and have included the photograph to show apparatus, etc.

I. *Procedure for New Method*

1. Into a 250-c.c. beaker (No. 2 low form) pour 5 assay tons (146 c.c.) of cyanide solution.
2. Add 20 c.c. of a 20 per cent. solution of lead acetate.
3. Add 15 c.c. of concentrated hydrochloric acid.
4. Place a 1/4-in. rod of zinc in the beaker (No. 1 in the photograph).
5. Place beaker on a hot plate—bumping does no harm if it will not break the beaker by raising it off the plate.
6. As soon as the solution boils, leave it so for 5 min.; then remove from hot plate.
7. Fill with cold water; then decant about half and again fill with cold water.
8. Twist the zinc rod quickly between the finger and thumb and draw it out of the sponge.
9. If any small particles of lead are left adhering to the rod at about where the top of the solution touched it, draw the sponge up the side of the beaker with a glass rod.
10. Touch the zinc rod to the sponge to free the particles.
11. Wash the sponge three or four times with cold water to remove zinc chloride.
12. Press it against the side of the beaker with a glass rod to remove the water.
13. Decant the water and wash again.
14. Place the dewatered sponge on a piece of sheet lead 2 in. square, then fold it. It is now ready for cupellation.

II. *Procedure When the Sponge Breaks*

When the sponge breaks into large pieces it is possible to unite them by pressing together with a glass rod. If there are many small particles it may not be possible to unite them by pressing together. Then this method is suggested:

15. Fit a 2-in. funnel into a filtering flask.
16. Cut a piece of sheet lead 3 in. square; fold as you would a filter paper.
17. Cut off the corners; then open it to fit the funnel.

18. Place about 5 g. of test lead in the lead cone and push it down with the finger, then smooth out the folds and creases so that it will fit well.

19. Lift it from the funnel and prick seven or eight holes in the apex or point by means of a pin, then put it back into the funnel.

20. Complete 9 and 10; then start the filter pump.

21. Tip the beaker and draw most of the lead sponge into the lead cone by means of a glass rod (without a rubber tip).

22. Pour the remainder of the solution into the cone, rinse beaker and wash contents of cone three or four times.

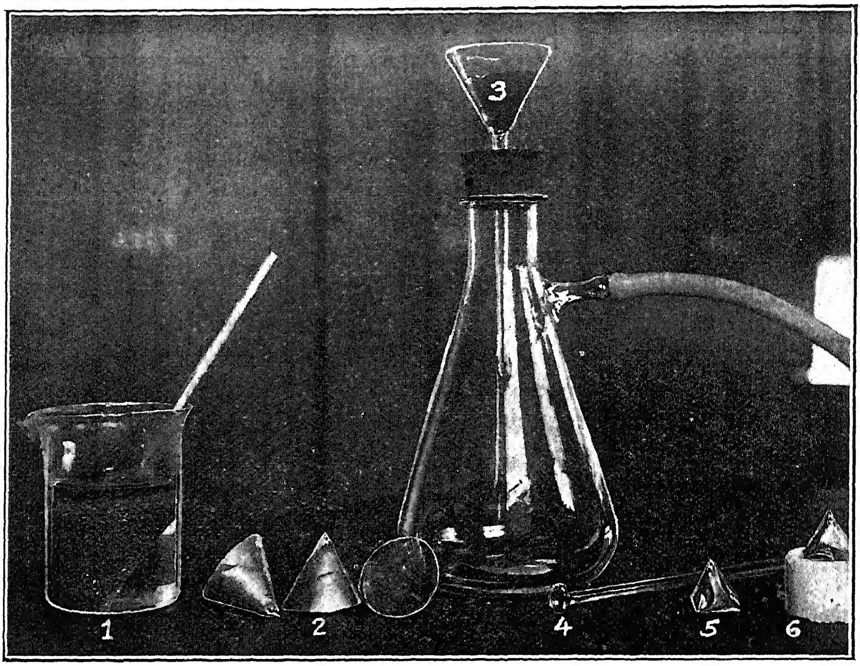

FIG. 1.—APPARATUS USED IN THE NEW METHOD FOR THE DETERMINATION OF GOLD AND SILVER IN CYANIDE SOLUTIONS.

23. Tamp the lead down with the flat knob of a glass rod. (See 4 in Fig. 1.)

24. Stop suction, remove cone from funnel and fold it in carefully. (See 5 in Fig. 1.) It is now ready for cupellation.

25. Draw a hot cupel to the front of the muffle and place the cone in it. (See 6 in Fig. 1.) When the water has been driven out, push the cupel into a hotter part of the muffle and finish the cupellation in the regular manner.

III. *Table Showing Amounts of Solution of Lead Acetate and Acid Necessary for Different Quantities of Solution*

Assay Tons	Equivalent in c.c.	Lead Acetate Solution, c.c.	Conc. HCL, c.c.	Time (including 5 min. boiling), Min.
5	146	20	15	20
10	292	25	25	34
15	437.5	30	40	35
20	583	45	50	42
25	729	50	75	44

IV. *Notes and Comment*

The generation of hydrogen along the zinc rod is sufficiently active to prevent the adherence of the lead. A strip of aluminum does not work so well.

The method has been used in the assay of a number of solutions containing a variety of base-metal compounds and in each case the sponge remained whole. With solutions from cobalt ores that contain much silver the sponge is apt to break.

The amount of time as given in the table will vary with the heat of the hot plate—those given are averages.

In order to keep the weight of the lead to be cupelled down to a minimum, thin sheet lead should be used.

A cone as described in 16 and 17 will correspond to a filter paper 7½ cm. in diameter and, made of heavy sheet lead, will weigh about 7 g.

The pin holes should always be as near the point as possible.

A screen analysis of the test lead used in developing this method is as follows: + 30, 7.0 per cent.; + 60, 28.6; + 100, 23.2; – 100, 41.2 per cent.

It is quite possible to have a cone manufactured as are bottle caps. This would lessen the weight about 50 per cent.

V. *Scorifying the Precipitate*

The following method may be used for scorifying the precipitate obtained by any of the methods of Group 3:

26. Collect the precipitate in a 9-cm. filter paper; wash it down into the point.

27. Make a cone from a disk of sheet lead 3½ or 4 in. in diameter.

28. Punch 10 or 12 holes at the point.

29. Fold the filter paper into a small wad and place it in the point of the cone.

30. Pour on top of the paper 10 g. of test lead.

31. Fold the lead cone so as to include its contents and place it in a glazed scorifier at the mouth of the muffle.

32. When the paper has become dry and begins to char, the gases will burn as they come from the pin holes. As soon as the flame ceases place the scorifier in a hotter part of the muffle. The lead cone will hold the paper firmly while it is burning; so there is no danger of its unfolding and scattering the precipitate.

Discussion

E. J. Hall,* New York, N. Y. (communication to the Secretary†).—The Chiddy method for determining gold and silver in cyanide solutions has been subjected to so many proposed modifications, of which L. W. Bahney's is one, it is natural to suppose the method is defective.

There is no denying that this method is capable of giving inaccurate results, but so is any other method of quantitative determination. However, proof has not come to my notice that the troubles are inherent and not in the detail application, and I am inclined to think from our experience that in many cases it is the latter.

Errors in this method may be occasioned by:

(1) The retention of zinc in the sponge, which tends to desilverize the molten lead in cupellation, similar to the action in the Parkes process, removing gold and silver in a crust of zinc oxide. Failure to remove zinc may result from lack of concentration of hydrochloric acid, due to neutralization by salts present, volume of liquid too great for the specified amount of acid, or the use of weak acid. (HCl rapidly loses its strength unless kept in perfectly tight containers. The HCl gas escapes even from the ordinary glass-stoppered acid bottles after the plaster seal has been broken.) Undissolved zinc is likely to be held in the upper part of the lead sponge, which floats above the liquid, and unless the solution is boiled it will escape action. Mr. Bahney's modification makes this difficulty impossible. It is not, however, a necessary evil when the assay is properly conducted.

(2) In the assay of foul solutions the sponge is apt to disintegrate, as pointed out. Occasionally this tendency may be overcome by slightly oxidizing the solution after adding zinc and boiling to reduce impurities. The lead acetate and remaining acid is then added. When the sponge breaks up it is usually sufficient to decant most of the liquid and then wash the lead into a boat, drain off remaining water, and cupel. If the lead is fine enough to remain in suspension, prohibiting decantation, it may be rapidly filtered on SS No. 597 filter paper. Fluxing is not necessary, as the paper may be pressed between blotters, sponge separated, paper folded once, held by one end with a pair of tongs and the bottom

* Department of Metallurgy, Columbia University.
† Received Mar. 28, 1915.

ignited with a bunsen burner, allowing ashes to fall into a lead boat containing the sponge; then wrapped and cupelled.

(3) The furnace temperature necessary to start cupellation is considerably higher than that required for a good finish. The major portion of cupellation loss occurs when the last gram or two of lead is being oxidized. This loss increases rapidly with increase in temperature, particularly with small beads such as are usually obtained from cyanide solutions.

There is about 1.7 g. of lead in 10 c.c. of a saturated lead acetate solution, the quantity usually employed. The maximum weight of sponge will not exceed this weight and is often less than 1 g. when the solution has been heated for a long time or HCl concentration is high. The sheet lead required for wrapping is 2 to 3 g., so the whole packet will not weigh over 3 to 5 g.

For the reasons given above it is impossible to obtain good results in cupelling lead buttons of this size unless the cupels are moved to the mouth of muffle or coolers introduced in the furnace immediately after the buttons have opened. This is particularly true when large cupels are used, as they give up their heat slowly. If gold is the principal metal sought it should always be protected by silver.

That gold and silver in all cyanide solutions can be best determined by any one method is not a reasonable assumption. For rich solutions where 1 to 2 assay tons will suffice and prompt returns are not required the evaporation method in a lead boat is most attractive, as the working time and attention are least. A lead boat 1.75 by 3.75 in. made from a sheet 3 by 5 in. will permit evaporation of 2 assay tons of solution in 60 to 75 min. A covering of test lead on the bottom of boat will facilitate evaporation and reduce the tendency to spit. If the solution is foul or carries suspended matter, scorification is desirable.

Mr. Bahney objects to this method because the lead foil may leak. Assayers' sheet lead holds water better than the foil and is readily obtainable.

Solutions that are reasonably free from impurities and suspended matter give good results by the Chiddy method and its modifications. Suspended matter not soluble in HCl will collect in the sponge, as well as other impurities, and scorification should follow. Under these conditions one of the so-called precipitation methods is preferable.

It is unfortunate that Mr. Bahney did not include some comparative figures, particularly as the assumption that some gold and silver might be retained on the zinc stick seems reasonable.

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Cost Factors in Coal Production

BY WILLIAM H. GRADY, BLUEFIELD, W. VA.

(New York Meeting, February, 1915)

FACTORS entering into the market value of coal are its grade, and the cost of labor, material, and capital. Reduction in these costs cannot be expected in the future, and it therefore follows that greater economy in their use must be accomplished if it is desired to hold the present markets; more tons of product per unit must be obtained in a given time from both men and material, and means devised for increasing the percentage of lump and domestic sizes.

This paper is presented for the purpose of inviting discussion and stimulating thought in regard to the influence of methods of procedure and plans of mining, upon the quality of coal, the economical use of labor, material, and capital, and the cost of production. Even the better methods of the present day leave much to be desired, and a review of the reasons and necessities for their use may be of value in pointing the way to more economical mining. The writer does not attempt to discuss untried methods of mining, but rather to state clearly and concisely a summary of the results obtained under the several methods of procedure that have been adopted.

In some instances a high degree of concentration has been effected, and comparisons of the results obtained with the results of lesser degrees of concentration form an interesting study. It is believed that much may be accomplished in improving the quality, and in the more economical use of labor, material, and capital, by *concentration*, and it is to concentration in particular that your study is invited.

The writer, who has occasion to visit many mines, most of which are under different management, has been inclined to the opinion that improvement has been retarded in many ways and that mining has not as yet been entirely freed from the early-day practices which were forced upon it.

Practices resulting in lack of concentration, to which reference will be made, are: The absence of robbing, necessitating the frequent interposition of barrier pillars; lack of system and of proper supervision, resulting in losses of life and coal. These practices could not well be

avoided years ago, for with coal on the railroad cars selling at practically the cost of production, the incentive was to mine only the coal which could be produced at a profit, regardless of what the future cost might be.

The writer, in talking with some pioneer operators in West Virginia, was much impressed to learn that in the early '80s, not only were pillars not robbed in some mines, but many of the operators firmly believed that the extraction of the pillars was a physical impossibility. Quite as much impressed were these operators when it was stated to them that to-day, in some mines, over 95 per cent. of the coal in the seam was being extracted, and that pillar coal, if properly mined, is of the same quality as room coal, and cheaper, both to the operator and the miner, to produce.

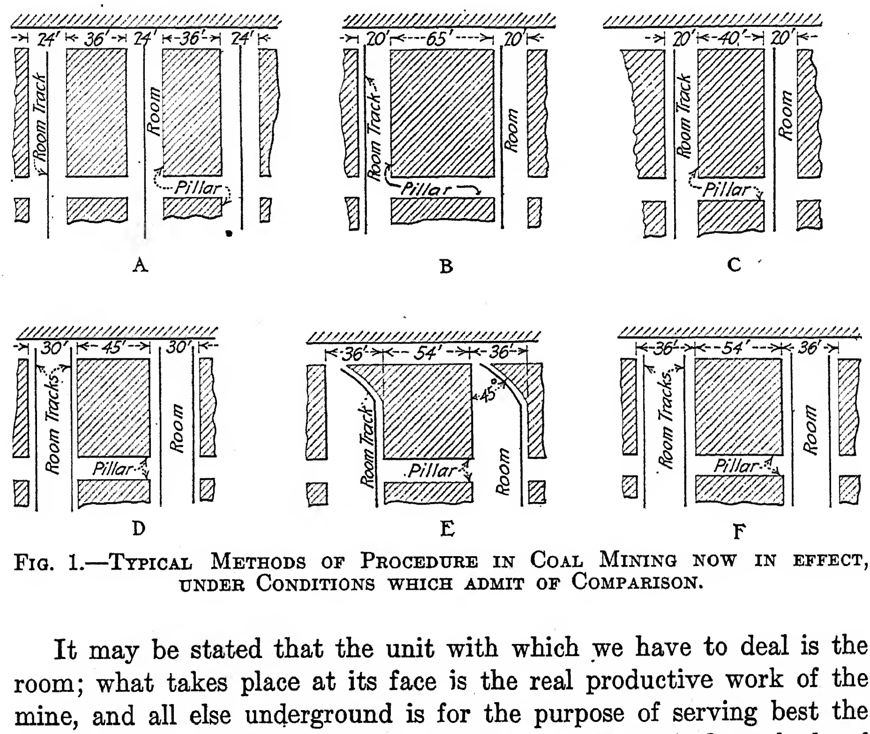


FIG. 1.—TYPICAL METHODS OF PROCEDURE IN COAL MINING NOW IN EFFECT, UNDER CONDITIONS WHICH ADMIT OF COMPARISON.

It may be stated that the unit with which we have to deal is the room; what takes place at its face is the real productive work of the mine, and all else underground is for the purpose of serving best the worker at the room face. Fig. 1 shows several typical methods of procedure that have come under the writer's observation. They are of particular interest in that one may see them in mines following the same plan, working the same seam, under conditions which admit of comparison. The features of these methods are given in Table I.

In all of these methods variations may be seen, from entries driving with no rooms turned to entries driving with two or more rooms turned and driving as the entries advance; in respect to the robbing, one may see variations from robbing following immediately upon the comple-

tion of the first two rooms to the robbing following at an indefinite date after the completion of the first workings of the panel. Where continuous paneling, or advancing robbing, is in effect, robbing is not compelled to wait until the completion of all the entries of the panel.

TABLE I.—*Methods of Procedure*

Sketches	A	B	C	D	E	F
Width of room in feet......	24	20	20	30	36	36
Width of pillar in feet.......	36	65	40	45	54	54
Location of track........	In center of room	Along robbing rib	Along robbing rib	Along robbing rib	Along robbing rib	Along robbing rib
Location of gob	Along both ribs	Opposite robbing rib	Opposite robbing rib	Between tracks	Opposite robbing rib	Between tracks
Number of men per room....	1 to 2 rooms	1	1 or 2	2	6	4
Feet of room face per man	48	20	20 or 10	15	8.5	9
Feet of entry per man....	120	85	60 or 30	37.5	15	22 5

The number of rooms per entry varies from about 12 to an indefinite number, and the depth of the room varies from about 300 to about 800 ft. The amount of timber and the manner and time of placing same depend largely upon the individual miner, and as a rule there are no specific instructions for his guidance; also, in general, no effort is made to recover the timber in robbing.

A method of procedure observed by the writer (but which has not as yet been sufficiently tested out in the matter of recovering the pillar to warrant its unreserved adoption), is shown in Fig. 1, *E*. Here it is intended that rooms shall be driven 36 ft. wide on centers 90 ft. apart, carrying a room face at an angle of 45° and a single track along the robbing rib but curved to parallel and follow the length of the room face. It is intended to work six men to the room, the gathering motor receiving and placing three cars at a time. Immediately upon the completion of the room the pillar is to be withdrawn. By this method of procedure a high degree of concentration will be effected and the efficiency of the gathering motors, mining machines, and miners will be increased. It is also hoped that by carrying the working face on a diagonal, fewer unexpected falls of top will occur than at present, because the fracture will generally be partly exposed before the entire coal support is removed from beneath it.

The relative degree of concentration effected in the above methods of procedure is shown in Fig. 2. The units of measure adopted for comparison are linear feet of room face per man and linear feet of entry per man.

In most mines the miner at the face is responsible for the safe working conditions of his room. In the above methods of procedure, therefore one might say that, for the same expenditure of time, energy, and watchfulness, the relative degree of security which the miner may feel as a result of his efforts is inversely proportional to the room space he occupies. It is also true that for cars of the same height the energy expended by the miner, or the work done in loading the coal, is much less where two or more men work per room and the room space per miner is low, than where one man works per room and the room space per miner is high.

In the grouping of rooms as outlined in Fig. 1 many arrangements were made from which have matured certain well-defined plans. Prob-

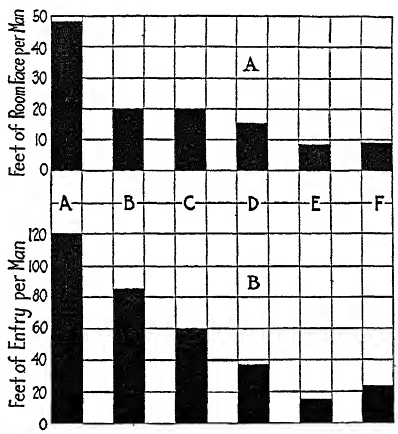


FIG. 2.—GRAPHICAL COMPARISON OF THE RELATIVE DEGREE OF CONCENTRATION IN THE METHODS OF PROCEDURE SHOWN IN FIG. 1.

ably the consensus of opinion favors the panel system, but even with it there are differences of opinion. Many men think that the entries should be driven to the inside lines of the property and the coal extracted retreating; others think that half of the property should be worked advancing and the remainder retreating; yet others think that all or nearly all of the coal should be extracted as the entries advance. It is the writer's opinion that the coal should be extracted in such a manner that the present worth on the returns from the mining venture will be greatest, both to the property owner and the operator, leaving only such coal during the advance of the entries as will permit of profitable mining until the final exhaustion of the property. Figs. 3 and 4 show typical plans on the panel system. Fig. 3 is the square or rectangular panel, Fig. 4 the continuous panel.

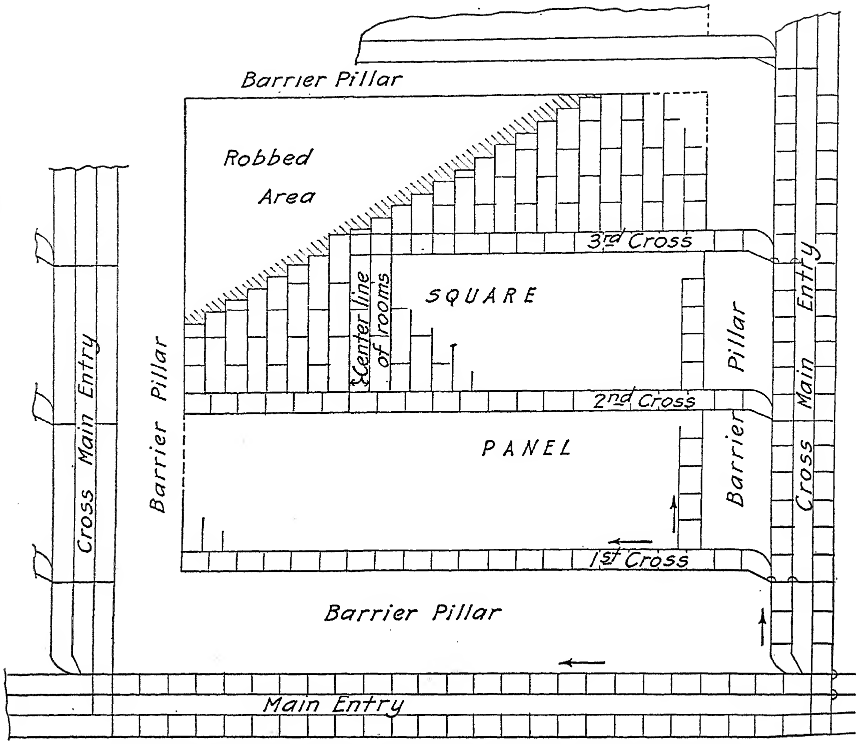


FIG. 3.—TYPICAL PLAN OF MINING ON THE PANEL SYSTEM.

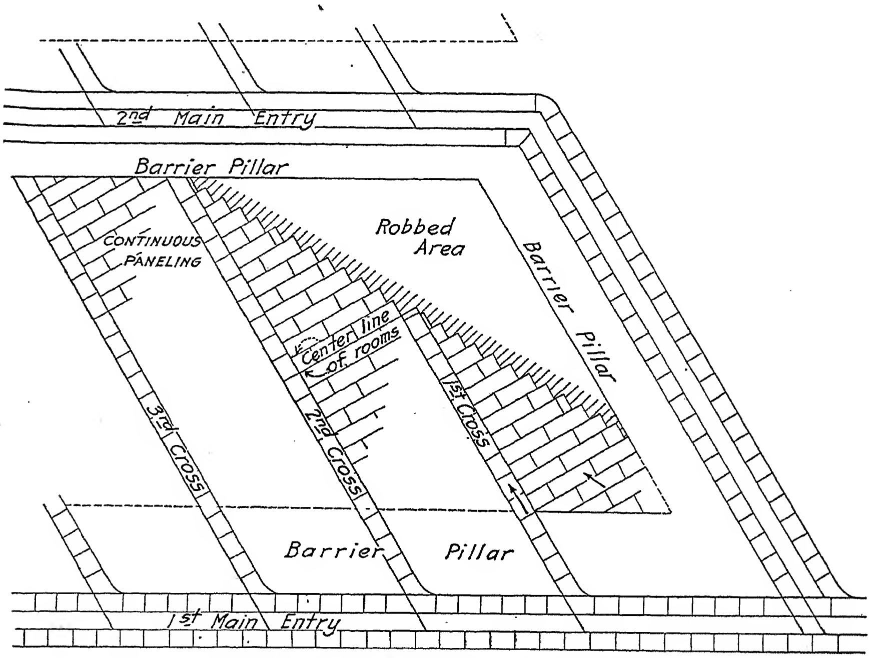


FIG. 4.—TYPICAL PLAN OF MINING ON THE PANEL SYSTEM.

Fig. 5 represents a typical lease or property of 1,000 acres from which it is desired to produce about 2,800 tons per day when running at a maximum. It is assumed for the purpose of this discussion that the coal is fairly clean, 6 ft. thick, and the condition of grades, top, and bottom fair. It is also assumed that the rate of loading per man per day is 16 tons. The questions to be decided are: What method of procedure and what plan are best? In order intelligently to make a decision the following information is desired:

1. What period of time will be required to reach the output?
2. How many day laborers, mining machines, mine cars, mules for gathering, and main-haulage motors will be required?
3. How much main entry, main entry track and trolley wire; cross main entry, cross main entry track and trolley wire; room entry, room-entry track, and rooms, and room track will be required?
4. What is the length of the average car haul?
5. What is the relative amount of power for ventilation?
6. What is the acreage of standing pillars, the estimated relative cost of production, and the estimated percentage of recovery?

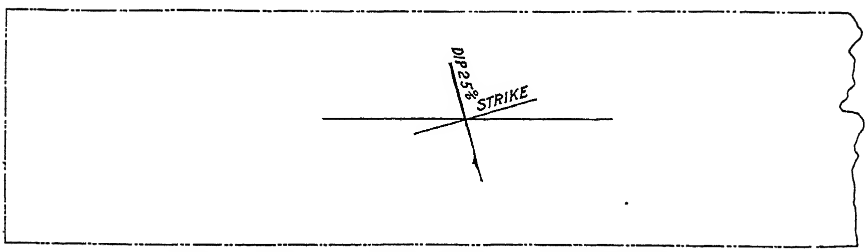


Fig. 5.—Plan of Property to be Developed.

It is not necessary to apply all of the methods of procedure to the development of the property under consideration in order to illustrate the thought in mind, and it has been decided to apply the methods referred to in Fig. 1, *C*, and the plans of mining in Figs. 3 and 4, as follows:

First Form.—Drive the third entry of the panel, turn the last two rooms on this entry first, start removing the pillar immediately upon the completion of the next to the last room, and continue to drive all of the rooms in the panel only fast enough to provide for the uninterrupted advance of the robbing. Work two men to the room and in the air courses and on the pillars. Only this method of procedure will be applied to the square and continuous panels, and the following methods to the square panel.

Second Form.—Drive the rooms of the panel as they are encountered, turning the first entry of the panel when it is reached, and start robbing immediately upon the completion of the last room on the third entry of the panel. Work one man to each working place.

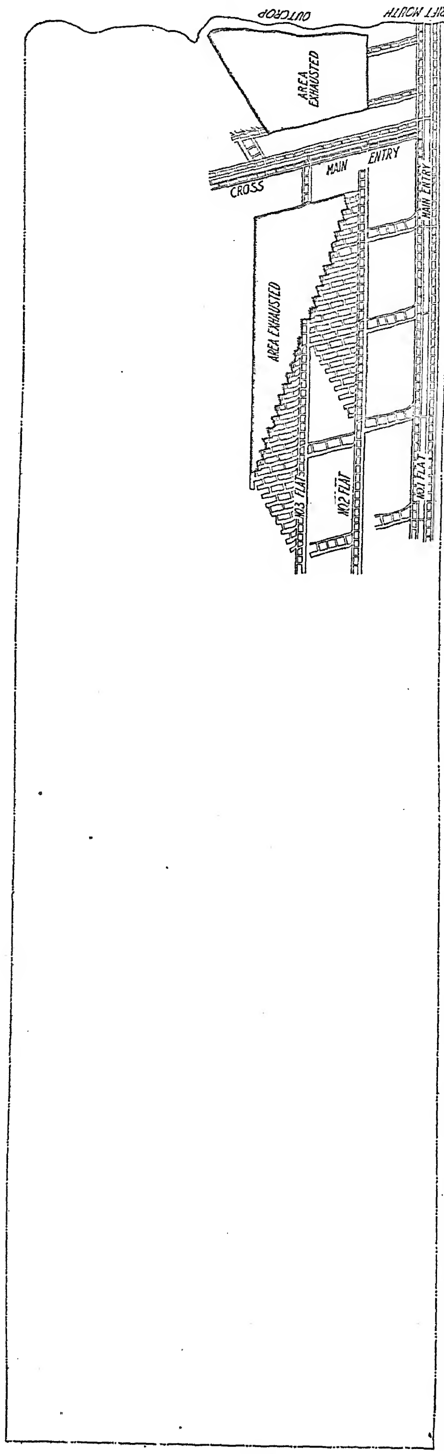


Fig. 6a1.—Advancing System with 4 Men to a Room. Rooms 36 ft. Wide, 54 ft. of Pillar. Position of the Workings at the Time the Output is Reached

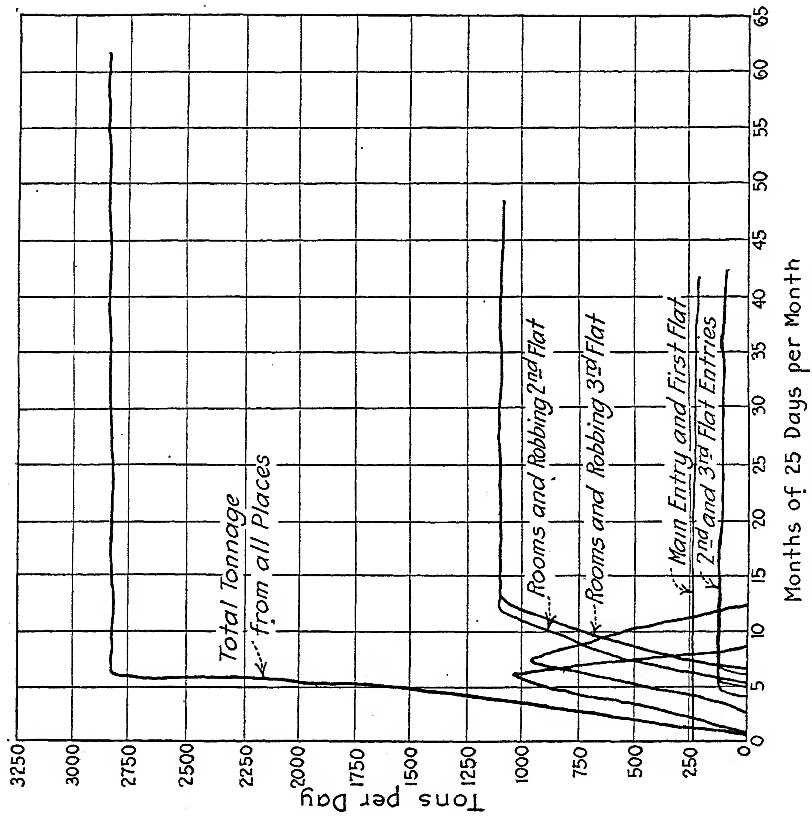


Fig. 6b1.—Tonnage Curves under the Method of Procedure shown in Fig. 6a1.

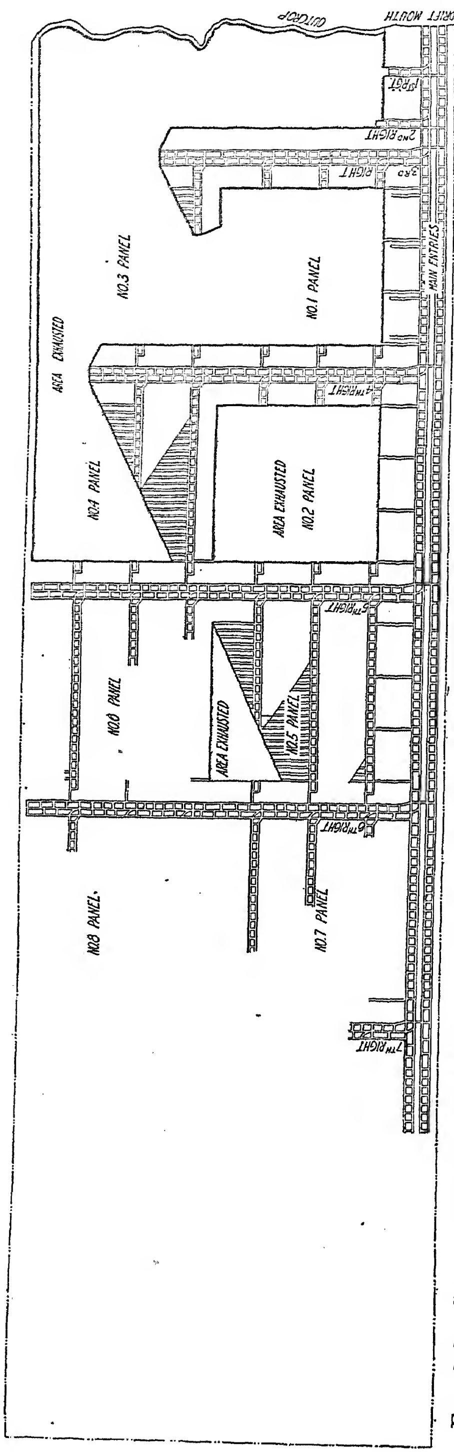


Fig. 6a2.—Square Panel with 2 Men to a Room, Robbing Retreating following Immediately upon the Completion of the Room. Rooms 20 ft. Wide, 40 ft. of Pillar. Position of the Workings at the Time the Output is Reached.

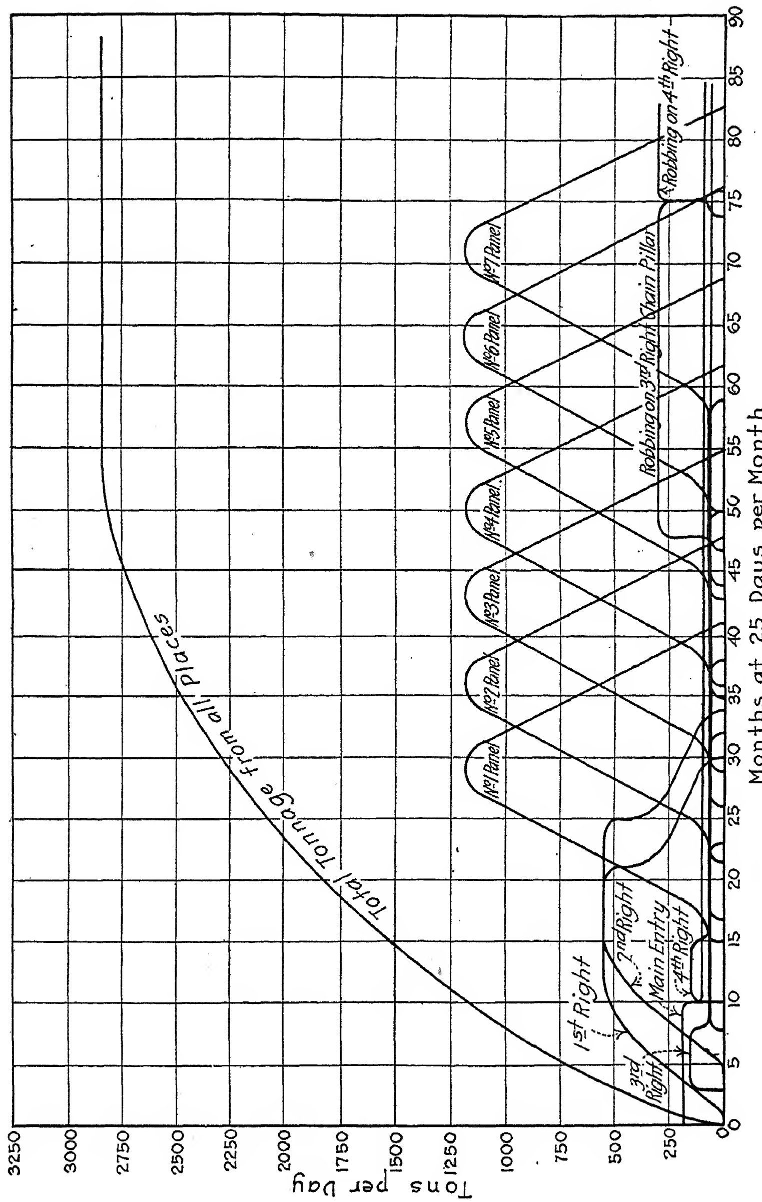


FIG. 6B2.—TONNAGE CURVES UNDER THE METHOD OF PROCEDURE SHOWN IN FIG. 6A2.

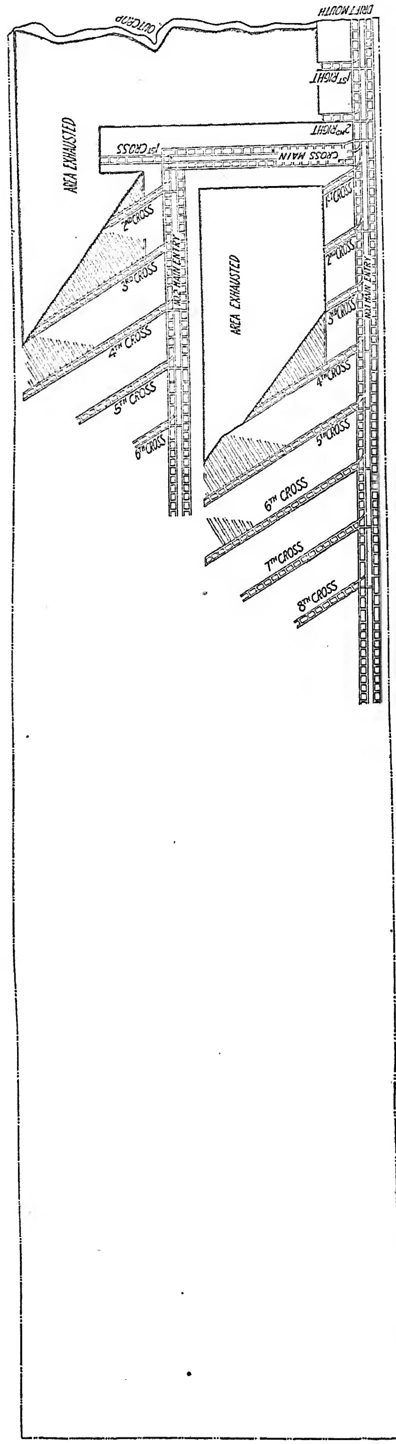


FIG. 6A3—CONTINUOUS PANEL WITH 2 MEN TO A ROOM, ROBBING RETREATING FOLLOWING IMMEDIATELY UPON THE COMPLETION OF THE ROOM. ROOMS 20 FT. WIDE, 40 FT. OF PILLAR. POSITION OF THE WORKINGS AT THE TIME THE OUTPUT IS REACHED.

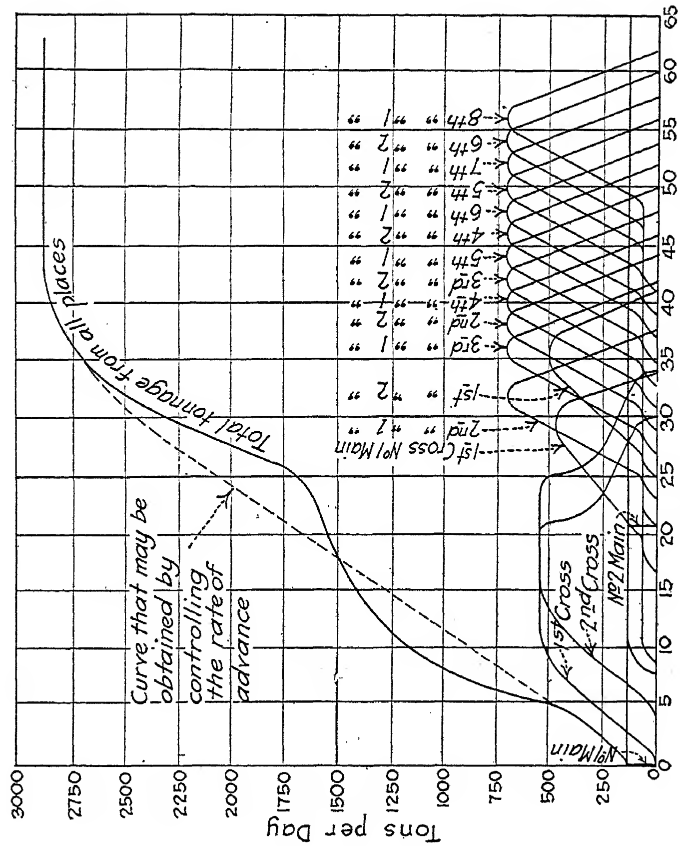


FIG. 6B3.—TONNAGE CURVES UNDER THE METHOD OF PROCEDURE SHOWN IN FIG. 6A3.

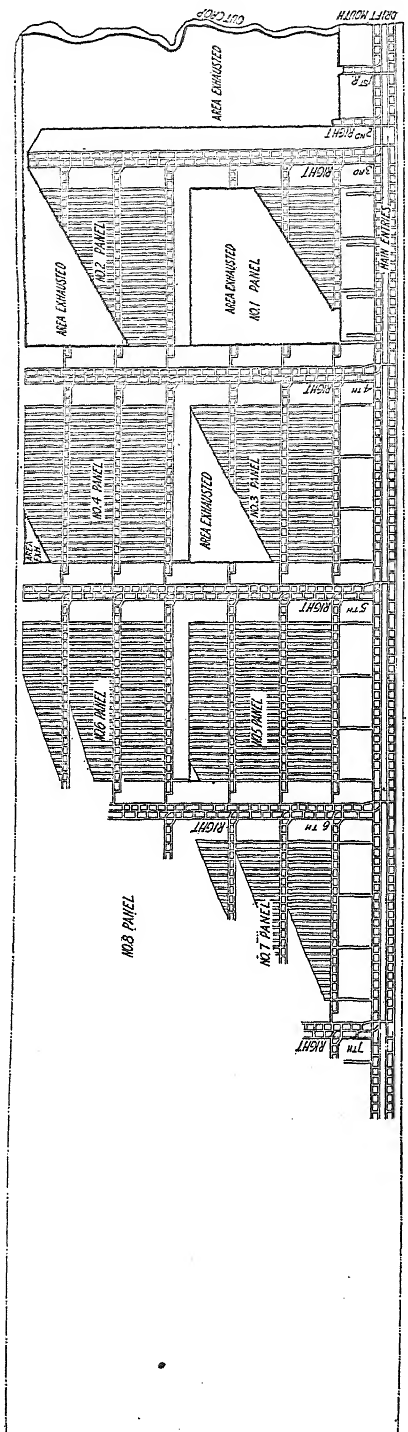


FIG. 6A4.—SQUARE PANEL WITH 1 MAN TO A ROOM. ROOMS ARE DRIVEN AS THEY ARE ENCOUNTERED AND ROBBING IS CONDUCTED RETREATING IMMEDIATELY UPON THE COMPLETION OF THE PANEL. ROOMS 20 FT. WIDE, 40 FT. OF PILLAR. POSITION OF THE WORKINGS AT THE TIME THE OUTPUT IS REACHED.

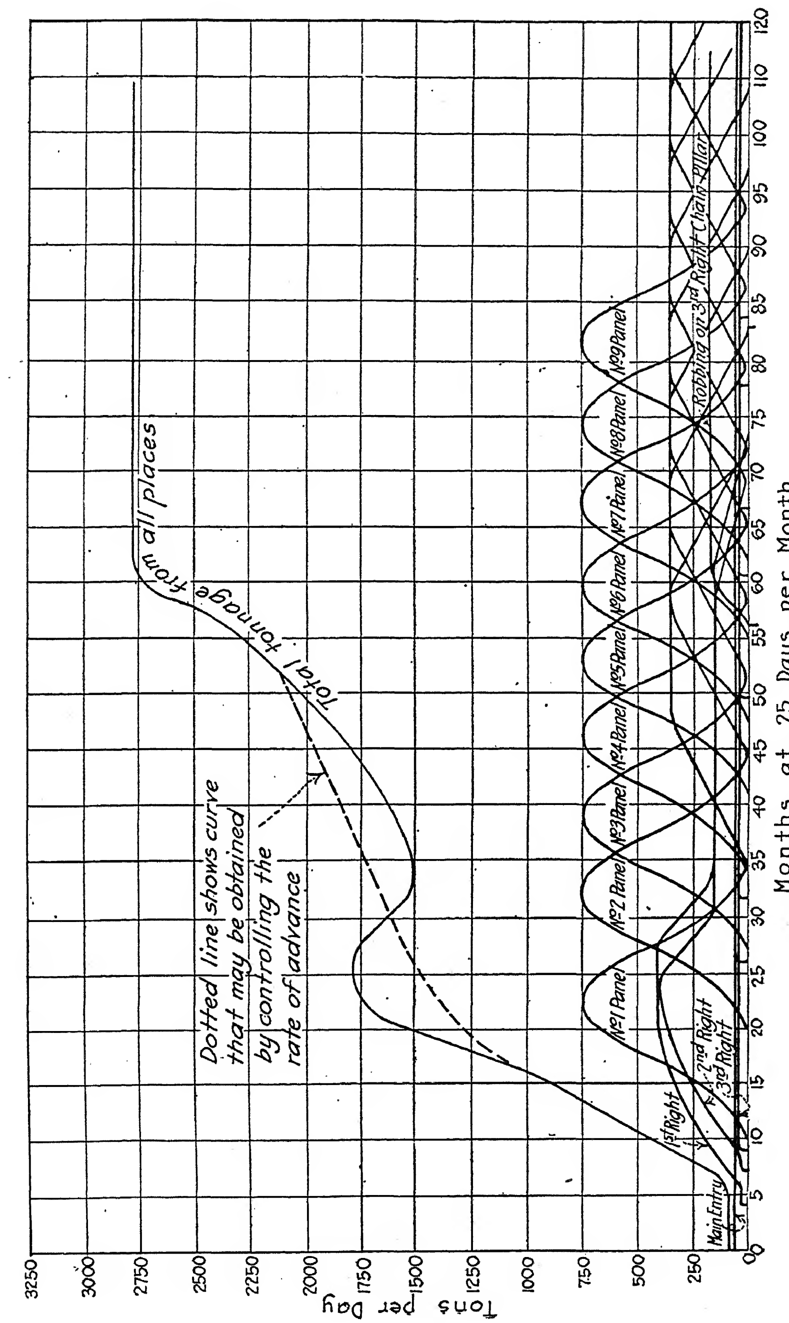


FIG. 6B4.—TONNAGE CURVES UNDER THE METHOD OF PROCEDURE SHOWN IN FIG. 6A4.

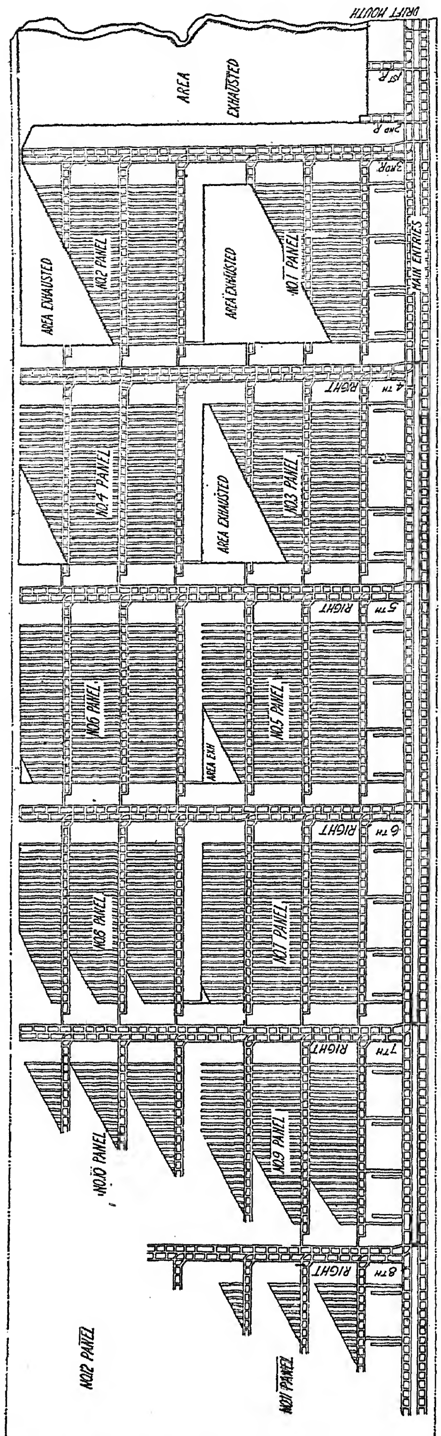


FIG. 6A5.—SQUARE PANEL WITH 2 ROOMS TO 1 MAN. ROOMS ARE DRIVEN AS THEY ARE ENCOUNTERED AND ROBBING IS CONDUCTED RETREATING IMMEDIATELY UPON THE COMPLETION OF THE PANEL. ROOMS 20 FT. WIDE, 40 FT. OF PILLAR. POSITION OF THE WORKINGS AT THE TIME THE OUTPUT IS REACHED.

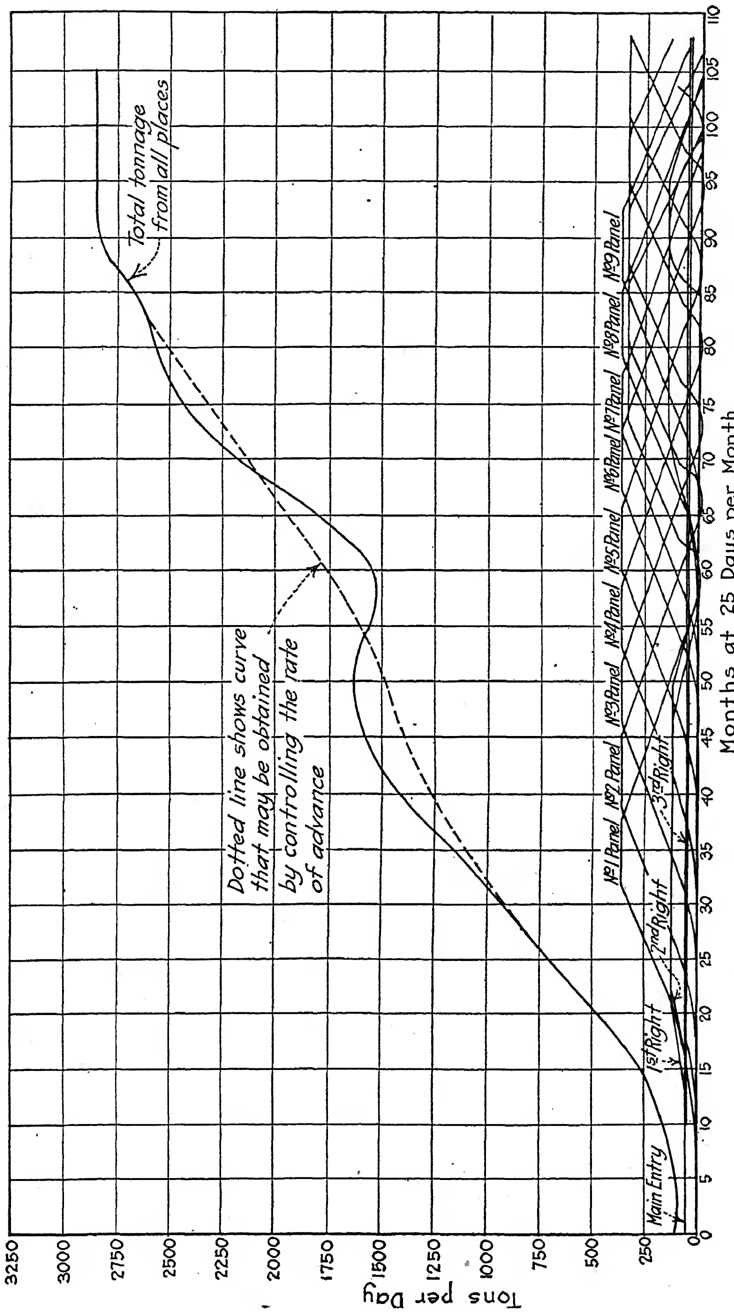


FIG. 6B5.—TONNAGE CURVES UNDER THE METHOD OF PROCEDURE SHOWN IN FIG. 6A5.

Third Form.—Drive the rooms and entries of the panel as they are encountered, start robbing immediately upon the completion of the last room in the third entry, and continue the robbing until the completion of the panel. Work one man to every other room, but advance all rooms and work one man to each pillar.

Table II shows the comparison, as well as some other figures to which further reference will be made.

TABLE II

	First Form	Continuous Panel	Second Form	Third Form	Advancing Method
Output reached, months	53	42	62	92	7
Day laborers	60	65	82	102	31
	Advancing method uses 8 asst. foremen.				
Mining machines	9	9	14	18	8
Mine cars	275	310	335	465	155
Mules	18	22	24	32	10
Motors	4	4	5	6	2
Main entry, Main-entry track, Main-entry trolley	7,850	6,500	9,300	13,950	600
Cross main entry, Cross-main-entry track, Cross-main-entry trolley	5,550	5,150	8,850	13,500	1,000
Room entry and room-entry track	10,700	12,700	33,900	50,400	7,000
Room track	15,840	20,500	96,800	230,300	18,100
Average car haul	6,180	5,333	7,420	10,230	3,640
Ventilation power, kilowatt-hours	40	42	125	175	20
Acreage of standing pillars	62.8	65.7	168.2	277.0	13.8
Relative cost of production	1.33	1.24	1.76	2.1	1
Percentage of recovery	94	95.5	83	80	97

From these data, it may be concluded that the first form of procedure and the plan of mining, Fig. 4, are the best.

The period of time required to reach the desired output was determined, for the several methods, as illustrated in Fig. 6, *A;* the location of the working faces from day to day, as determined by the assumed rate of advance of 16 tons per man, was plotted on a map, and the total number of faces at the time the desired output was reached were counted, from which data the tonnage curves were plotted as shown in Fig. 6, *B.*

In arriving at the relative number of men and mules required, the writer used certain rates of performing certain tasks taken from time-study observations. The amount of rolling stock required is based on the assumption that the equipment will travel at the same rate of mileage per day; the other items compared were taken direct from the maps. In the absence of facts for comparison the writer used his opinion, based on observation, and the opinions of managers now operating under the plans compared.

All of the above methods of procedure and both plans of mining have been designed to meet certain wants. In some instances certain features of the plan have been prescribed by the land owners in order to safeguard their interests from "squeezes" and losses of coal due to lack of proper supervision. Were the proper supervision supplied and better methods of procedure adopted, the restrictions in the plan of mining might very properly be removed. Other details of design have been the result of accepting certain "rules of thumb" which have since been proved wrong, and yet other details, although admittedly wrong and expensive, have been introduced rather than combat the wrongs which they are designed to circumvent.

In the plan of mining shown in Fig. 3, the frequent interposition of barrier pillars is for the purpose of confining a squeeze and limiting its range of destructive action. The use of these barriers is imperative under the methods of procedure that involve large areas of long-standing pillars and where the degree of supervision is low. It is to be regretted that their use is so common, for they tend to interfere seriously with the maximum degree of concentration because one is seldom, if ever, able to provide a satisfactory output from a single panel, and then only for a short period of time. Where two or more panels are required to produce the output, the further the workings advance the more distantly separated they become, or other important considerations must be sacrificed. Disadvantages of the unit-panel plan may be seen in the curve in Fig. 7, which shows the great variation in the tonnage obtained daily, varying from zero at the opening of the panel, augmented by a more or less constant rate of increase, to a certain maximum number of tons, and then a gradual decline to zero again. If a certain number of tons per day gathered from the panel is accepted as 100 per cent. efficiency for a gathering motor, as shown in Fig. 7, it will be noticed that the motor is at first working at a very low efficiency, which gradually increases until the maximum is reached, at which time another motor must be added, and the average efficiency of the two motors is about 50 per cent.; there is a similar drop in efficiency with each motor that is added, until the maximum tonnage from the panel is reached, after which the process of removing motors from the panel is begun. In some measure this degree of efficiency may be increased by working the motors over more than one

panel, as is often done, and a better efficiency curve might be obtained more nearly in accordance with the full line shown, but in practice a rigid watch must be kept on this detail, or more often than otherwise a lower degree of efficiency than that shown will result.

If, in the preceding paragraph, instead of considering the efficiency of the gathering motor the efficiency of day laborers or the tons produced per unit of material and equipment in use had been considered, the same general discussion would apply. It is the writer's observation that where low efficiencies are obtained from day laborers, material, and

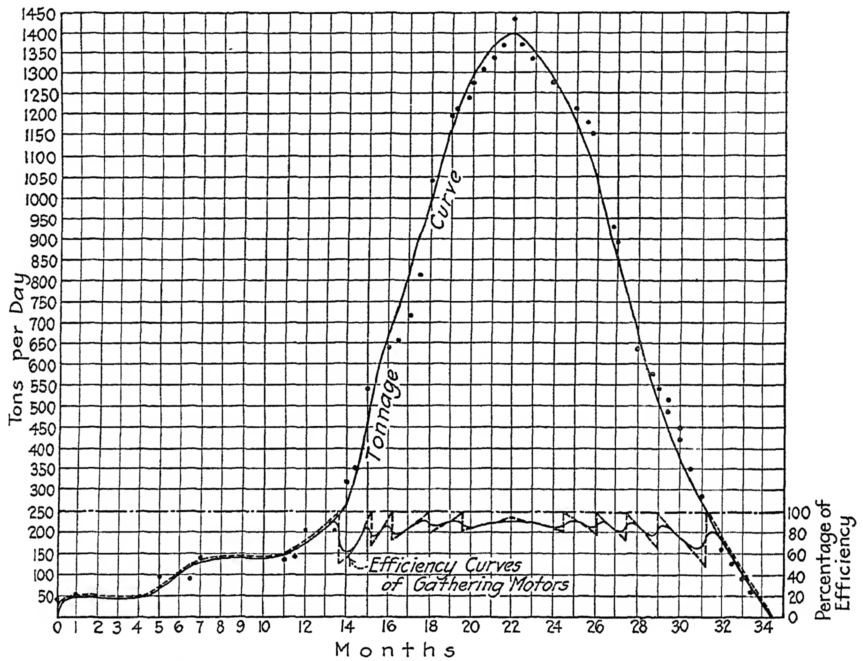


FIG. 7.—CURVES SHOWING THE VARIATION IN THE TONNAGE DAILY OBTAINABLE FROM THE UNIT PANEL WHEN PROCEEDING AS OUTLINED IN FIG. 6A2; ALSO SHOWING THE EFFICIENCY OF GATHERING MOTORS WORKING IN THE PANEL.

equipment, low efficiencies are also obtained from the miners at the room face. For these reasons it is difficult, and in practice well-nigh impossible, to establish any constant relation between a given tonnage desired to be uniformly produced, and the amount of material, equipment, and day laborers required to produce that tonnage; the efficiency of these quantities rises and falls with the rise and fall of the tonnage curve, although in an erratic manner.

Thus it would appear that the square panel, while designed to meet certain requirements, does so at the loss of much that is to be desired, and introduces new complications. The barrier pillars are, as the

term implies, for the purpose of barricading against some impending danger, such as an unforeseen squeeze. Since no one can predetermine where or when these squeezes will occur it sometimes happens that barrier pillars are provided where they are not needed, and are omitted where they are needed; yet experience has shown the wisdom and necessity of their use under certain conditions. They would be used less frequently if the square panels were made rectangular, but the same degree of security would not be obtained unless the entries were driven to the limit of the rectangle, with few or no rooms driven as the entries advance. If we accept it as axomatic that when a room is driven to completion its pillars should be immediately removed in order to obtain the best results, or that it is equally as fundamental to open up no new entries until ready to mine from them, and that mining should then be conducted at the maximum rate of production, the rectangular panel that involves either long-standing pillars or long-unproductive entries must be rejected.

The continuous panel obviates the necessity for frequently interposing a barrier pillar and it is especially well adapted to a property where the main entries are driven to the dip. However, the tonnage from a single continuous panel is limited, and where the main entries of the property go to the rise the maximum degree of concentration cannot be obtained or the rooms off of the cross entry will go to the dip. Advancing robbing is impracticable because the pockets in the pillars go to the dip. The rate of production from a single room entry rises and falls in the same manner as the rate of production in the room entries of the square panel and the general discussion above in reference to the square panel applies to the continuous panel.

However, if one follows the history of the development of mining methods from the early-day single-entry system to the present-day panel system, it will be found that the square or nearly square panel meets sound mining practice more closely than any of the plans which have preceded it. Until methods of procedure are adopted which make the restrictions of the panel unnecessary, or until a plan of mining is devised without the objectionable features of the panel, but retaining its many favorable features, the square panel will be accepted by many operators as the standard plan of mining.

For many years it has been the common belief that coal could be most economically cut and blasted by using a depth of cut equal to the height of seam. This erroneous idea frequently resulted in blasting down more coal than could be loaded in one day and not enough for two days or for two men in one day, and was the cause of allotting more than one room to a miner. That the height of seam does not bear any direct relation to economical cutting or blasting was demonstrated by the United States Coal & Coke Co. at Gary, W. Va., working with a Sullivan

shortwall machine, and the writer's observations are that miners are pleased to work two or more to a room, provided their earnings are as great as when they work in rooms by themselves.

Much thought is being given to the subject of mining methods. Probably the most marked results have been achieved by the officials of the above-mentioned company. They realized the objections to the mining methods outlined above and applied themselves to working out a plan which would be simple, direct, and efficient. They accepted it as axomatic that any change in the prevailing plans of mining must be beneficial to the property owner, operator, and miner alike, for any change that would benefit one or more of the interested parties at the expense of the others would not last.

In this study difficulty was experienced because of the entire lack of systematized knowledge as to the proper relative rate of advance of room to retreat of pillar, the most economical width of room, and in fact what might be considered 100 per cent. efficiency for any man, animal, or machine about the mines. In order to determine these data, which were absolutely essential to an intelligent solution of the problem, a series of time studies was instituted and extended over a period of weeks, covering all of the motions that make up certain underground operations that have to do with getting the coal from the working face to the railroad car. Thousands of observations were taken, properly checked, tabulated, collated, and used as a basis for a method of procedure, which has been put to the rigid test of practical use with remarkably good results.

This method of procedure has for its object the maximum degree of safety, sanitation, and opportunity to the miner, and of security to the property owner, while at the same time offering the greatest advantage to the operator. It combines a maximum degree of concentration with a minimum of expenditure for labor, material, and equipment, in such a manner that these *quantities bear a constant relation to the output.* Its use has resulted in a marked reduction in fatalities, increased earnings to the miners, decreased costs per ton for labor, material, equipment, and capital, and the recovery of practically all the coal in the seam.

At Gary, W. Va., mules are used for gathering, and as a result of concentration their efficiency has, in some instances, been increased over 200 per cent. At one of the mines, fewer day laborers are employed underground than are employed about the tipple.

For the purpose of comparing the results obtained under this method with those from the several methods of procedure in the panel system, the writer has applied the method to the property and the problem under consideration. Fig. 8 shows the arrangement of the workings at the time the desired maximum output is reached. It also shows the details of the method of procedure; the other data desired are given in

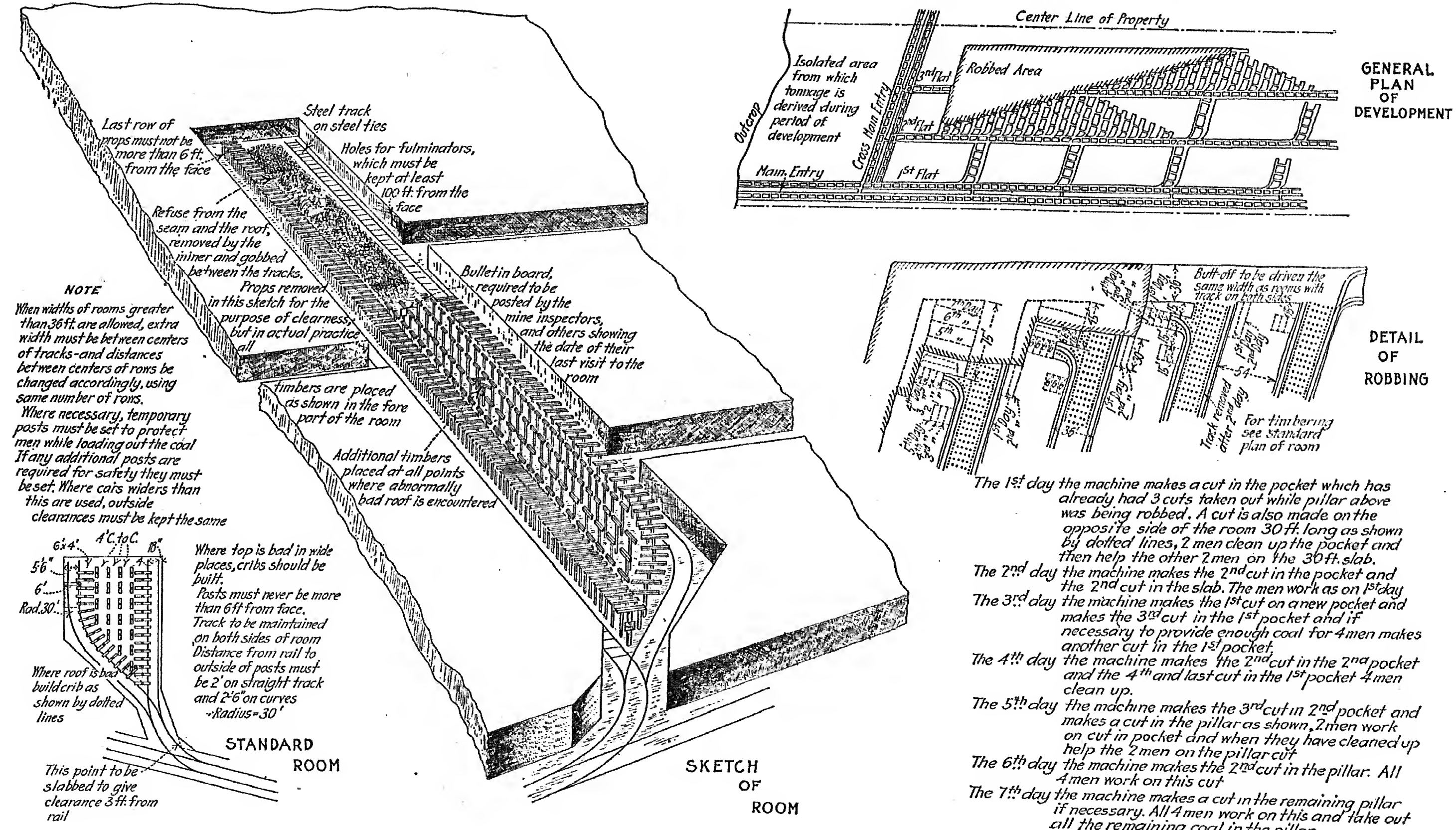


FIG. 8.—ADVANCING SYSTEM OF MINING, WITH ROOMS 36 FT. WIDE AND 400 FT. LONG. UNITED STATES COAL & COKE CO., GARY, W. VA.

Table II. Fig. 9 shows the tonnage curve and, for comparison, the total tonnage curve from Fig. 6B2. The tonnage curve, Fig. 9, from the unit entry shows that the tonnage rises very rapidly until the maximum is reached and *then continues indefinitely at that rate of production.* By using available data, the proper length of room, angle of breakline and angle of advancing faces may be predetermined, so that the total daily tonnage from the entry is a multiple of the tons that can be hauled

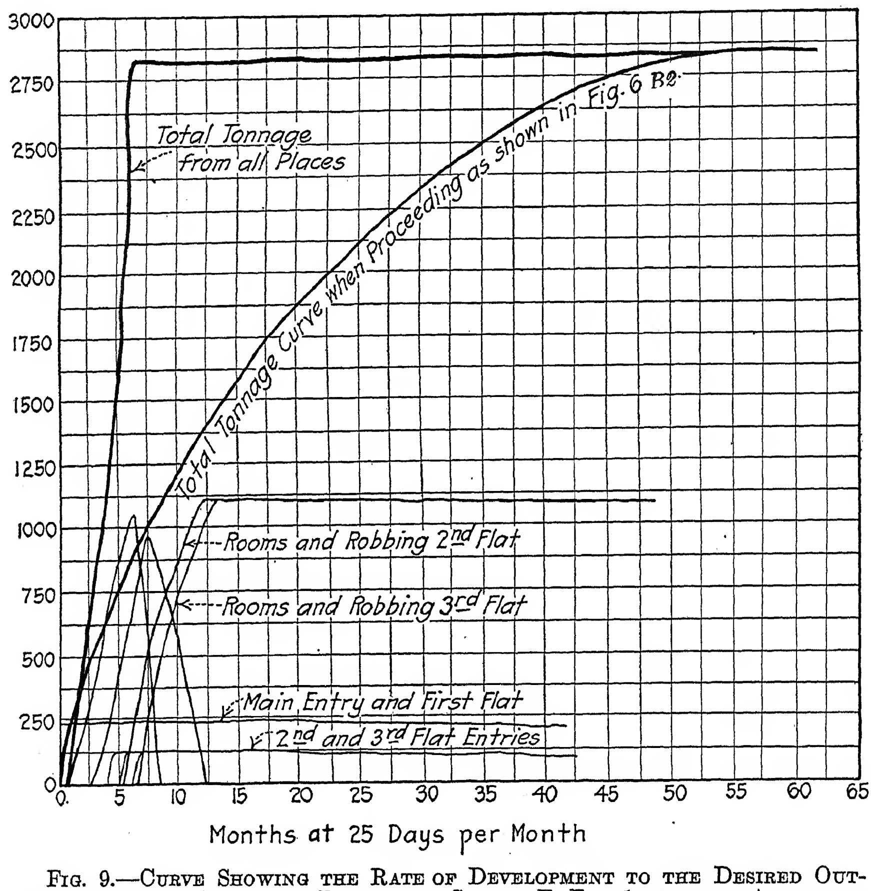


FIG. 9.—CURVE SHOWING THE RATE OF DEVELOPMENT TO THE DESIRED OUTPUT, UNDER THE METHOD OF PROCEDURE, SKETCH F, FIG. 1, AND THE ADVANCING PLAN OF MINING, FIG. 8. ESTIMATED PERCENTAGE OF RECOVERY, 97 PER CENT.

by a mule or motor; thus, the mules or motors are always working at maximum efficiency. It is equally true that when the workings have advanced for a short distance, after reaching the maximum tonnage from the entries, the estimated minimum number of day laborers required may readily be confirmed, and once the entry reaches its maximum tonnage, and the quantities of labor, material, and equipment have

been accurately determined, these quantities remain constant throughout the entire extent of the entry, which may be as great as the property is long.

Fig. 2 shows that the room space occupied per miner is less than in any of the other methods now in effect, which is an index of the relative degree of safety a miner obtains for a given expenditure of time and energy. The excellent manner in which the rooms are timbered, shown in Fig. 8, is the minimum required; where the mine foreman or miner has reason to believe that additional timber is required to make the place safe, the miner must place additional timber before doing anything else. That these precautions and the high degree of supervision exercised are worth while may be seen from the paper by Howard N. Eavenson.[1]

As the entries advance, all rooms are driven and robbed immediately upon their completion, and rooms are opened up only fast enough to provide for the uninterrupted advance of the robbing. Thus no barrier pillars are required, for the virgin coal protects the workings on three sides and the weight of the roof is resting on the bottom in the robbing. If a disturbed area of coal is encountered, or for some reason it is desired to discontinue the panel, a barrier pillar may be introduced at any time exactly where it is needed and the entries continued for the purpose of exploration.

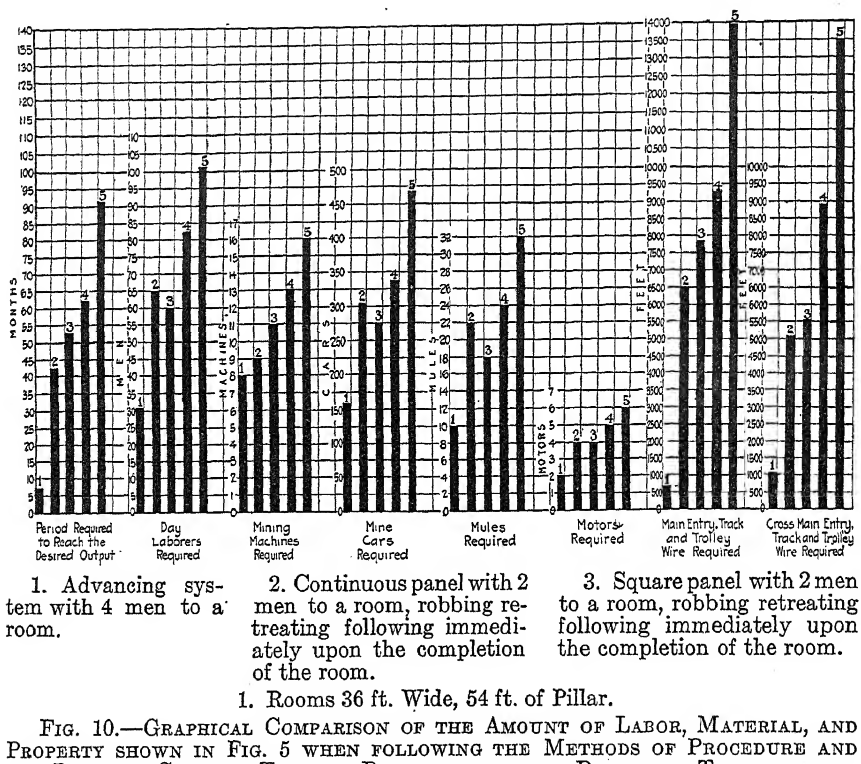
In order that the different methods of mining may be readily compared, Fig. 10 was prepared showing the relative amount of labor, material, and equipment required to produce the tonnage desired from the property shown in Fig. 5; also the acreage of standing pillars, the relative cost of production, and the estimated percentage of recovery.

The writer has given much time and study to the subject of this paper and concludes that any method of procedure that does not provide for the removal of pillars immediately upon the completion of a room is fundamentally wrong, because it involves long-standing pillars open to the unfavorable influence of atmospheric agencies and other forces of nature; the duplication of track work; the cleaning up of many slate falls that might otherwise have been avoided; and the scattering of workings, all of which increase the cost per ton for labor, material, and equipment, and cause the pillar coal to be badly disintegrated and low in domestic and lump sizes.

It sometimes happens in practice, however, that fundamentals must be sacrificed to adapt the method to peculiar conditions encountered, often resulting in lack of concentration and large areas of standing pillars. Where considerable tonnage is desired and a new property is being opened, skilled miners, experienced in robbing pillars, are hard to

[1] Safety Methods and Organization of United States Coal & Coke Co., this volume, p. 319.

get and frequently the officials, mine foreman, and underbosses are not experienced. In order to keep up the tonnage under these conditions, the workings must necessarily become distantly separated, because coal can only be obtained from room workings. It frequently happens also that the rates for mining pillar coal and room coal are not properly adjusted, so that the men can earn more in room work than in pillar work, naturally causing the pillars to lag behind, and requiring the introduction of barrier pillars to safeguard against squeezes; these barriers in turn cause a further separation of the workings, and a decrease in



1. Advancing system with 4 men to a room.

2. Continuous panel with 2 men to a room, robbing retreating following immediately upon the completion of the room.

3. Square panel with 2 men to a room, robbing retreating following immediately upon the completion of the room.

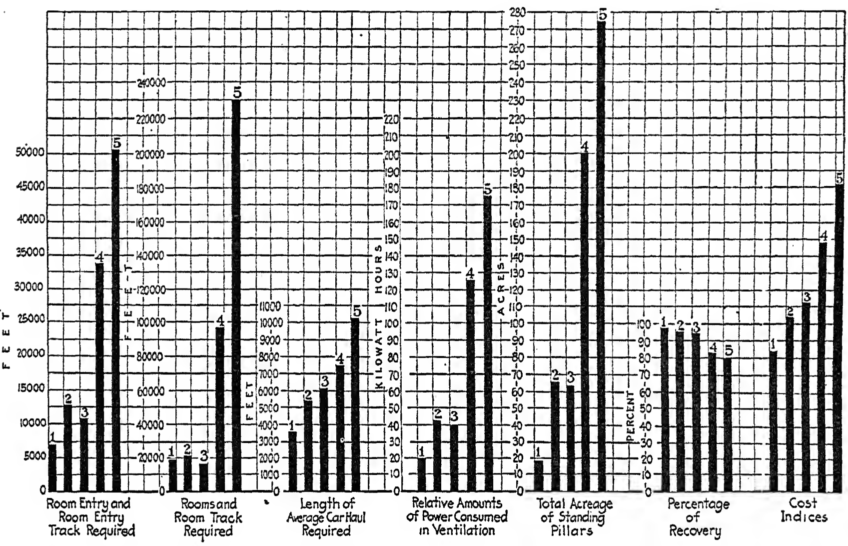
1. Rooms 36 ft. Wide, 54 ft. of Pillar.

Fig. 10.—Graphical Comparison of the Amount of Labor, Material, and Property shown in Fig. 5 when following the Methods of Procedure and the Relative Cost per Ton, the Recovery, and the Period of Time required

the efficiency of labor, material, and equipment. The natural impulse of the mine foreman, therefore, is to open up more rooms in advance of the robbing in order to increase the efficiency to something like a proper standard.

For these reasons the territory for a given output during the development period should be as isolated as possible, and no greater in extent than is practicable. After the development period is passed and the organization perfected, in the opinion of the writer, there is no good reason why a mine operation should not be conducted with much the same regularity as a blast furnace or an industrial railroad.

The fallacy that the average miner will load only so much coal and no more has long since been exploded, and it is a matter of every-day observation that miners are pleased to load coal if the mine cars are given to them with some degree of regularity and with some relation to the time required to load a car. Not long since the writer observed the tally sheets at one of the mines visited, and was pleased to note that of 132 miners loading that day, the average tons per miner per day was over 250 per cent. more than the average of the State. When one considers, however, that a coke loader, working under the heat of the sun and of



4. Square panel with 1 man to a room. Rooms are driven as they are encountered and robbing is conducted retreating immediately upon the completion of the panel.

5. Square panel with 2 rooms to 1 man. Rooms are driven as they are encountered and robbing is conducted retreating immediately upon the completion of the panel.

2, 3, 4, 5. Rooms 20 ft. Wide, 40 ft. of Pillar.

Equipment required to produce an Output of 2,800 Tons per Day from the Plans of Mining outlined above. Also the Acreage of Standing Pillars, to reach the output.

the coke ovens, will load from 35 to 40 tons as an ordinary day's work, there is no reason why a miner working under so much more favorable circumstances should not load at the same rate. In this connection the following observations that have to do with loading coal underground are interesting.

These figures show that less than 47 per cent. of the time spent underground was consumed in loading coal and over 12 per cent. of the time was lost waiting for the empty mine cars. It may be stated further that these men were loading at the rate of 35 tons per day of 8 hr., and actually did load at the rate of 16 tons per man per day per year.

Thickness of Seam, Feet	Loading Coal, Minutes	Waiting on Cars, Minutes	Resting, Minutes	Length of Shift, Hr. Min.
5.50	220	58	10	7 42
4.67	193	60	13	7 16
6.17	221	65	13	7 36
6.58	221	65	13	7 36
6.33	220	57	10	7 35
6.17	220	57	10	8 09
6.00	221	65	13	8 16
6.92	221	65	13	9 00
5.33	221	65	13	7 57
6.00	198	60	13	7 02
5.00	193	55	10	7 04
8.08	231	32	13	7 06
Average..........	214	59	12	7 40

The results obtained by the United States Coal & Coke Co. under the advancing system as compared with those obtained under the several other methods of procedure noted above may seem unreasonable and beyond expectation, but they fulfill the anticipations based on the data available when the method of procedure was formulated. Better results, in industries generally, invariably have been found to follow the presentation of opportunities to workmen, concentration, regularity, and the elimination of what may be termed lost motion, as shown by the marvelous increases in efficiency obtained by Gilbreth, Taylor, and other pioneers following the same line of thought. All of us remember how Gayley startled the metallurgical world by the introduction of dry-air blast, eliminating the periods of lost motion, which permitted of regularity and concentration in the operation of the blast furnace and produced results which at that time were almost beyond human credulity.

But in every instance where high records of efficiency have been obtained, it should be particularly noticed that the foreman and workmen were carefully following a plan, all the details of which had been carefully worked out by the management.

A detailed plan of mining and a projection with written instructions should be worked out by the management for every mine, and the mine foreman and laborers should not be permitted to deviate from same without the written consent of the management.

Curves and graphs should be plotted, showing an adopted 100 per cent. efficiency for the men and equipment, and curves should be plotted on the same chart showing what is actually being accomplished, as illustrated in Fig. 11, which is a time chart of a gathering locomotive under working conditions found in the mine, the summation of which is given in Fig. 11, *A*.

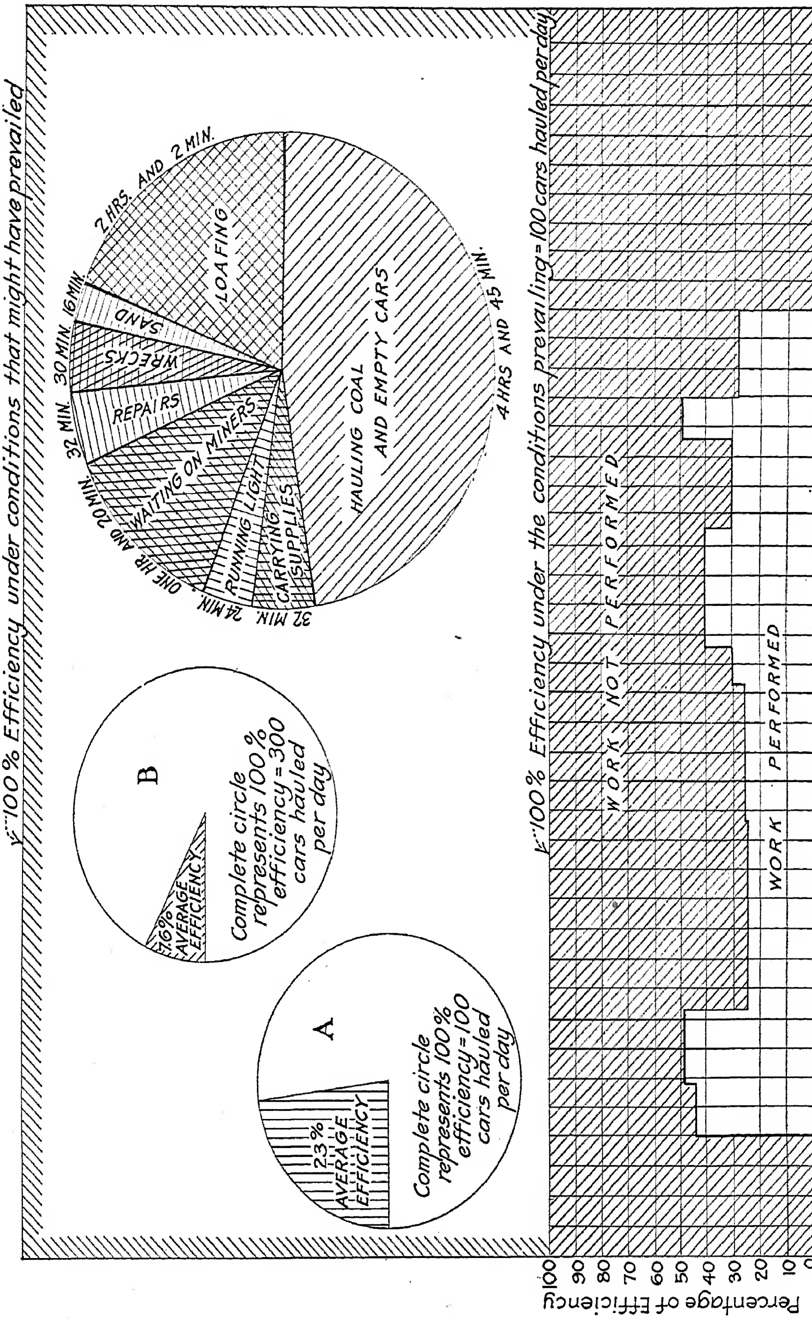


FIG. 11.—EFFICIENCY CHART OF A GATHERING LOCOMOTIVE UNDER CONDITIONS OF ACTUAL OPERATION

Whether or not it is practicable for operators working under any one system of mining, or any method of procedure with a given system of mining, to introduce another system or method of mining, the writer does not presume to discuss. The assumption is, of course, that each operator is doing what he considers best.

It might be inferred, however, from the example mentioned above that concentration and motion studies can be applied only where changes in the plans of mining are effected, or for the purpose of showing inefficiency. Many examples of constructive results might be mentioned and it is desired to call attention to Fig. 12, which is a reproduction of a mine now operating. It may be noticed that the mine is progressing

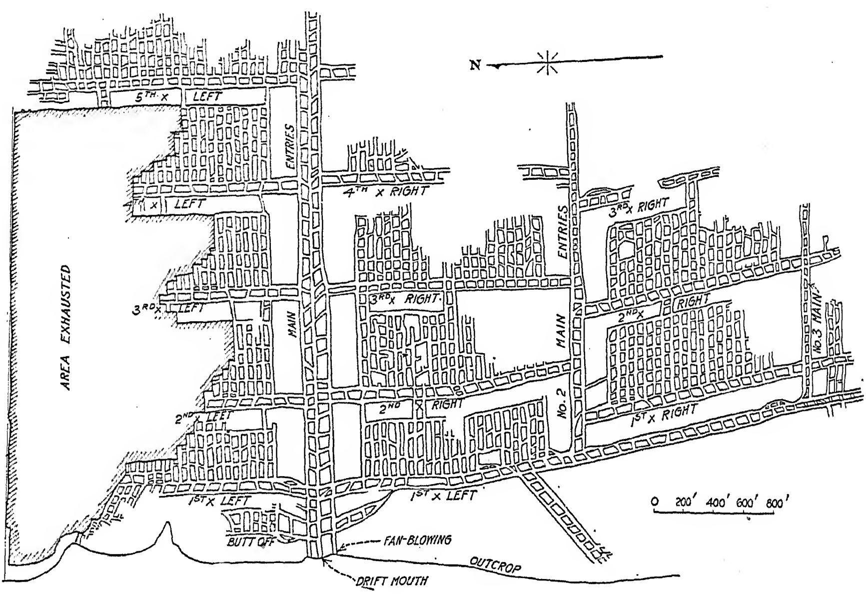


Fig. 12.

on a rectangular panel system, with workings off of No. 1, 2, and 3 main entries. Low efficiencies were obtained throughout the mine as a result of the widely separated condition of the workings and it was concluded to suspend operations on the No. 2 and 3 main entries, concentrating the men in the 1, 2, 3, and 4 left cross entries. It is emphasized that no change in the plan of mining was involved, but concentration, effected practically over night, resulted in a 50 per cent. decrease in the main-line motors, 47 per cent. decrease in the day laborers, and 30 per cent. decrease in the number of mules. The daily output was increased to the highest figure in the history of the mine and the daily earnings of the miners were also increased.

Changes in the method of procedure and plan of mining are now being

made, *without involving capital expenditures,* from which it is anticipated a very much greater degree of concentration will be effected, resulting in decreased cost of production, increased percentage of recovery and higher daily earnings to the miners, while at the same time reducing the amount of labor, material, and equipment in use.

The purpose of this paper will have been accomplished if it arouses an interest in the study of methods of procedure in their relation to the cost of production, and tends in the slightest degree toward safety and the conservation of human lives and coal resources.

In conclusion, the writer desires to thank Thomas H. Clagett, Chief Engineer of the Pocahontas Coal & Coke Co.; Edward O'Toole, General Superintendent, and Howard N. Eavenson, Chief Engineer of the United States Coal & Coke Co.; and James Elwood Jones, General Manager, Pocahontas Consolidated Collieries Co., Inc., for data and criticism in the preparation of this paper; and Warren T. Russell, Assistant Mine Inspector of the Pocahontas Coal & Coke Co., for assistance in the preparation of the figures.

Discussion

S. A. Taylor, Pittsburgh, Pa.—In connection with the conditions that exist in the Pocahontas field, is not the question of lump coal less of an item in the sale of the product than it is in some other sections? Take for instance the Pittsburgh section, we would be embarrassed considerably by the use of what you have termed the continuous panel, from the fact that it does not take into consideration the cleavage of the coal, and our output would contain a greater amount of fine coal when operating under the continuous panel than under the square panel.

William H. Grady.—In the Pocahontas field cleavage is not as highly developed as in some other sections of the country; for that reason it does not exert much of an influence in laying out the mine projection. It is also true, as you say, that the question of lump and prepared grades of coal has only recently exerted an important influence in the Pocahontas field. But to-day everybody is striving for a higher percentage of lump and prepared sizes because of the new markets, and the uses to which Pocahontas coal has been put, and the decreased demand for beehive coke.

It is generally admitted to be the better practice to take advantage of the cleavage, but if in attempting to do so it becomes necessary to drive workings to the dip or conduct robbing up hill, it is then a question whether or not the increased percentage of prepared sizes is worth more than it costs; if so, then undoubtedly it is proper to drive to the dip and rob up hill. If, however, you are able to drive the rooms on the faces, you will notice that only the faces of the main entries are adversely located, and

with machine cutting, that is not a very serious disadvantage as compared with the advantage of having the grades in your favor.

But generally speaking each mine is a problem in itself and just what is the best thing to do can only be determined by a careful study of the mine.

George S. Rice, Pittsburgh, Pa.—Fig. 3 indicates a system, but I see no indication of how the chain pillars and entry barrier pillars are withdrawn. There seems to be a very large amount of coal in the chain and barrier pillars.

William H. Grady.—The chain and barrier pillars are taken out when it becomes no longer necessary to hold the entry open as a haulway. The entries as shown in Fig. 3 are in use as haulways, but in Fig. 6A2 you will notice that in some instances the panels are completed and the chain and barrier pillars are being extracted.

H. M. Crankshaw, Lansford, Pa.—What is the best percentage of extraction in advance work before robbing is started?

William H. Grady.—The percentage of extraction, or the ratio of the width of the room to its pillar, in first workings depends on many factors and in general it will be found that the economical use of labor and material is the limiting factor, rather than stratigraphy. The amount of superincumbent strata, and their nature, the thickness of the seam and its nature, the nature of the immediate top and bottom; what constitutes the conditions under which a miner may work most economically; the economical use of labor and material; how long the pillars may stand, and what falls if any may be expected in the rooms during the period intervening between the first workings and the robbing—these are some of the more important factors to be considered in determining the proper ratio of room and room pillar. Probably as important a factor as any is the nature of the stratum immediately overlying the seam, or the "nether roof," as it is called. If this stratum is either very weak, say 30 in. of draw slate that falls on exposure to the air, or very strong, say gray sandstone, larger pillars will be required than otherwise in order to provide for the proper recovery of the pillar.

Where high recoveries and low cost are obtained, the present practice is to use room widths one-fourth to two-fifths of the distance from center to center of the rooms. Many mines have come under my observation where the room width was much greater; the costs were high and the percentage of recovery was from 50 to 75 per cent. A table has been prepared by the writer showing the limits beyond which the ratio of room width and pillar width must not go, for rooms 18 to 40 ft. wide, widths varying at intervals of 4 ft., and for covers from 100 to 1,200 ft. thick, the thickness varying at intervals of 100 ft.; which might have been introduced as part of this

paper, because the question enters into the cost of production, but it was concluded not to burden further the discussion.

H. M. Crankshaw.—As far as concentration is concerned, do you consider instances of two men employed in one room?

William H. Grady.—Concentration may be effected in almost every detail of mine work, but where two or more men are employed in a room a greater degree of concentration is possible. It is considered good practice to work as many men in a room as controlling factors will permit.

H. M. Crankshaw.—At a mine I have in mind, the coal was about 8½ ft. thick. Two men were worked per room, the size of the cut and the width of the room being so arranged that about 40 tons per day were shot down. The shooting was done after hours. This allowed about 20 tons per day per man. Do you know of any better conditions of work than this?

William H. Grady.—Twenty tons per man per day has been the average output for several years right along.

H. M. Crankshaw.—These figures were taken for a period of three months.

William H. Grady.—Was that the average output of the entire mine?

H. M. Crankshaw.—No, I am referring to one particular panel.

William H. Grady.—The figure that I have used—*i.e.*, 16 tons per man—is conservative. It is a figure obtained not for one particular section of a mine, or for one month, but has been obtained over a period of time. I find that the average of several mines, which come under my observation, was about 16.4 tons per miner per day, for the year 1914. Much depends upon the opportunity presented to the men; you will find practically everywhere that men are loading at a rate of from 30 to 40 tons per day, and I have no doubt but that miners could load 25 tons per day of 7 hr. if the opportunity were presented to them and the conditions were suitable. To-day little attention is paid to the matter of fatigue or to some of the most simple, though inexorable, laws of nature that have to do with a man's performance.

H. M. Crankshaw.—Are mules being used for hauling the cars?

William H. Grady.—Yes.

H. M. Crankshaw.—Is it your opinion that it is more economical to use the mule for haulage where the grades are favorable?

William H. Grady.—The question of mule haulage vs. motor haulage is an old one. There are conditions about some mines that preclude the possibilities of economical mule haulage and in all mines there are conditions where the motor will do from 10 to 40 times the work of a mule; but likewise in many mines there are conditions where for the same capital expendi-

ture for mules and the same operating expenditure for labor and material, the cost per ton for gathering will be less with mules than with motors.

The reason is obvious; under favorable conditions mules will travel under load all day at the rate of about 2.5 miles per hour, whereas under the same conditions motors will travel at the rate of about 5 miles per hour, but in handling the same load as the mule under the same conditions the motor cannot be under load more than half the time. It takes two fairly high-priced men to operate a motor and one fairly low-priced man to drive a mule, and the cost of cable alone sometimes equals the cost of maintaining the mule. The capital expenditure for the motor is about 10 times that of the mule. But unfortunately in practice sufficient attention is not paid to differentiating between mules and motors. Often you will find mules working where motors should be, and *vice versa*.

George S. Rice.—You speak of recovering 97 per cent. Is that theoretical or is it an actual figure?

William H. Grady.—The problem under consideration is a hypothetical one, and the figure to which you refer is an estimate, but it is based on figures taken from actual practice. Not long since I had occasion to figure the percentage of recovery from about 780 acres that had been mined, over 300 acres of which had been final mining, and the average percentage of recovery was above 94 per cent.

George S. Rice.—The percentage of recovery depends, of course, upon what you estimate to be the thickness of the seam. I do not see how you could get so complete a recovery with any system; you lose some pillars, do you not?

William H. Grady.—The method used to calculate the percentage of recovery is as follows: Accurate surveys are made, not to exceed 90 days apart, from which the area is determined by plotting and planimetering same. Sections of the seam are taken at such intervals as are necessary to obtain an accurate average section, and the volume is calculated by multiplying the average section by the area. Usually the total acreage exhausted in any mine will be divided up into small units of about 1 acre and the error between the actual volume excavated and the calculated volume will be small. The tonnage of these volumes is readily determined by applying the proper constants and the total tonnage, divided into the actual tonnage delivered from the mine, as recorded at the mine mouth, gives the percentage of recovery.

A method of calculation for determining the percentage of recovery sometimes used is the ratio of the acreage mined divided by the acreage mined and lost. This method is obviously wrong because of the varying section, but it could not be used in many mines of the Pocahontas field because the area lost cannot be measured in acres, because: (1) the area lost is seldom in excess of a few square feet in any one loss, and (2) there

is no one there to keep account of the area and the number of losses at the time the loss occurs.

George S. Rice.—That is a very good showing for any mine. In my own experience, even with longwall mining, I was not able to get more than 97 or 98 per cent., though that somewhat depended on what you estimate or assume to be the thickness of the vein. My observation down in the Pocahontas field is that some coal is lost by sticking to the rejected bone coal, besides what may be lost through crushing or covering up in pillar withdrawal.

William H. Grady.—In my opinion percentages of recovery are not absolute. But if you apply the same method of calculation to 75 or 100 properties and you find that some are obtaining 75 per cent. recovery, while others are obtaining 95 per cent., you are justified in saying that one is obtaining a 20 per cent. higher recovery than the other. Or if you are making these calculations from year to year in connection with a given property you are able to make fairly accurate comparisons.

Edwin Ludlow, Lansford, Pa.—In the advancing system of mining to which you refer, do you use no panels?

William H. Grady.—The policy is to open up from one to three entries, as the condition of the mine will permit, sufficient to give the maximum output desired (varying proportions of the maximum output may be obtained by changing the angularity of the robbing lines and the lines of advancing faces, or the number of men worked per room). Rooms are opened up only fast enough to provide for the uninterrupted advance of the robbing, so that you have good robbing falls behind, virgin coal on three sides, and a very limited acreage of active pillars in the immediate vicinity. Thus the possibilities of a squeeze are most remote and in case one should start it can readily be controlled.

Some of the advantages claimed for panels are that in the event of a catastrophe its range of destructive action is limited. This is wrong from two points of view. Catastrophes will always be with us so long as the line of defense is to limit their range of destructive action, rather than to eradicate their cause, and as a matter of fact, panels do not limit their range of destructive action. However, gas and dust accumulate in standing pillars, and by referring to Fig. 10 the relative possibilities for accumulations of gas or dust in the advancing system as compared with other systems may be seen.

Panels introduce numerous barrier pillars that might otherwise be avoided, and from the analyses in the paper it appears that their elimination is very much to be desired.

Edwin Ludlow.—You have a main haulage entry, do you not? What barrier pillars are provided there?

William H. Grady.—In the example shown in Fig. 8 the main haulage is through 1st flat; 1st, 2d, and 3d flats are continued to the extreme inside of the property. The solid coal flanking 1st flat, while serving in the capacity of a barrier pillar, is really acreage held in reserve, so that at the time the chain pillar of 1st flat is removed there will be sufficient coal tributary to the entry to provide a tonnage that may be readily mined without experiencing an increase in the cost of production or a decrease in the output.

George S. Rice.—There is one phase I am interested in inquiring about. I have not been in the Pocahontas field for several years, except for the investigation of several unfortunate disasters, but several years ago I paid a good deal of attention, while carrying on extensive mine sampling with a party in the Pocahontas field, to the matter of waste. I observed that under the systems then in vogue there was a considerable difference in the quality of the coal as loaded, between that from the faces and that from the pillars, apparently for this reason: The bone coal is easily rejected in loading in the advance workings; but in the older pillars the layer of bone, which is next the roof, was crushed down and so mixed with the coal that it was not possible for the miner to separate or pick it out. For this reason I imagine that it is an advantage to take out the pillars immediately.

William H. Grady.—That is very true, in my opinion. I do not have comparative figures over a long period of time, but I am reliably informed that there is a very marked increase in the percentage of prepared sizes, and at one place, where there is a washer working, an increase in the car yield is reported, due, I take it, to the decrease in the amount of slate mixed in with the pillar coal.

Pillars I have no doubt deteriorate some if they stand for years, and in many instances pillars are being mined to-day that are from 5 to 8 years old.

George S. Rice.—It was so in the case I spoke of. Do you not lose some of the curtain wall, or shell of the pillar next to the gob or robbing fall?

William H. Grady.—Yes, some of that shell is lost in many instances. If the robbing falls are not following closely upon the removal of the stump of the pillar, and the rooms are relative free from waste, the quantity of coal lost in the shell is very small; but where robbing falls follow quickly and the rooms are dirty many people use coal as a protection, and in order to avoid loading up slate. A better and cheaper practice is to set a row of protecting timbers.

The point you have brought up emphasizes the value of wide rooms and wide pillars, where conditions will permit of their use. You have fewer curtain walls and less loss of coal. In other words, the perimeter of the pillar is less per ton of coal in the pillar.

Edwin Ludlow.—Do you pull the timber?

WILLIAM H. GRADY.—The United States Coal & Coke Co. pulls the timber and the recovery is from 15 to 85 per cent., depending on conditions. But in general the timber is not recovered.

E. W. PARKER, Washington, D. C.—What was the average production per man of the United States Coal & Coke Co. as compared with the rest of the State?

WILLIAM H. GRADY.—I am not in a position to give you those figures, Mr. Parker. I judge that the tons per miner per working day is about 250 per cent. of the average of the State.

CHARLES ENZIAN, Wilkes-Barre, Pa.—I am certain that Mr. Grady does not desire his statement concerning anthracite methods to convey the impression that such are the result of modern design. For the information of those who are not familiar with the conditions in the anthracite region, the present difficulties experienced in the recovery of pillar coal are an inheritance of the effect of old-time mining methods, rather than our own liking, or present mine design.

WILLIAM H. GRADY.—It was not my idea to speak disrespectfully of the anthracite operators at all. I used the anthracite field as an illustration because of the large number of anthracite men present, and the wasteful practice with which we are all so familiar in the Schuylkill and Hazleton regions.

The paper by Mr. Whildin[2] at a recent meeting of the Institute shows very clearly that some anthracite operators realize very fully the enormity of the waste and the great benefit to be derived from concentration and having control of the robbing.

GEORGE S. RICE.—I would like to ask a question that has a bearing on the final recovery: that is, what percentage is permitted, by your company, in the advancing workings where the pillars are not immediately pulled? Also, what cognizance is taken of the depth of cover over the mine in figuring the percentage that may be taken out in advance workings?

In order to point the question, I refer to a matter relating to a very serious accident which occurred in Oklahoma a while ago, where so much had been taken out on advance workings in a steep pitching bed, that finally, when the cover became 750 ft. deep over the workings, the entire mine collapsed. I believe the Frick company of Pennsylvania has certain rules, but do not know whether they are invariably followed, but I understand it is its general practice not to take out more than 30 per cent. on advance workings. The company expects to take out the rest on the recovery of the pillars. Of course, the percentage would depend to a great extent on the depth of cover. The particular concern and interest is that it is a problem that arises from time to time in the West, particularly in mountainous districts.

ARTHUR HOVEY STORRS, Scranton, Pa.—Is it not probable that the

[2] Steep Pitch Mining of Thick Coal Veins, *Trans.*, l, 698 (1914).

question, as to the proper amount of the seam to be taken out in the first mining, is one to be considered entirely independent of the thickness of the cover? The matter of so proportioning the area mined out to the pillars that the largest percentage of the seam may ultimately be won, at a minimum of total cost, is, I believe, of more importance than the fixing of the sizes for the pillars simply to carry safely the overburden until time for the final robbing. The size of pillars which will give the best ultimate return from the seam is, I believe, in all cases, much in excess of the size of pillars required for the mere support of the overburden until time for final robbing.

So far as I have known it, in the anthracite fields, the removal of from 30 to 35 per cent. of the seam in the first mining results in the ultimate recovery of the largest possible proportion of the pillars and at a minimum cost, and this regardless of the depth below the surface of the coal seam. This is particularly true where the workings stand for a number of years before robbing is commenced, for they will undoubtedly be found in better shape for the robbing work if the original mining has taken only a small proportion of the seam. Of course, if it is not expected that any robbing of pillars is to be done, then the matter of the calculation of the best sizes of pillars to be left for proper surface support is an essential one.

George S. Rice.—Actual safety is what I had in mind.

William H. Grady.—I take it that what Mr. Storrs has in mind is this: The limiting factor in the proportioning of room to room pillar will be found not to be the thickness of the cover, but rather the economical use of labor and material, and it will usually be found that what is a proper extraction in this respect falls inside of the extraction allowable so far as superincumbent strata are concerned.

As a general proposition, I think Mr. Storrs is correct: *i.e.*, the limiting percentage of extraction on the advance workings is not determined by the amount of cover; but where the "nether roof" is very weak, or where it is very strong, 30 to 35 per cent. extraction in the advance workings precludes the possibilities of high recoveries unless the pillars are immediately removed. Also where the depth of cover is very great the percentage of extraction in first workings must not be as much as 30 per cent.

George S. Rice.—One more point I might mention, an extreme case that came under my observation while doing some work for the Santa Fé R. R. some years ago. The Starkville mine of Colorado is working toward Fisher's Peak, the front point of a *mesa* south of Trinidad, the top of which is nearly 3,000 ft. above the coal bed. When the cover became over 1,500 ft. in depth a squeeze started, and it was found impossible to hold up the roof in rooms. They were obliged to drive to the boundary of the panels, turning no rooms; then starting at the boundary drive up the rooms and pull them back quickly. The roof and floor are both strong sandstone;

and even in the advance entries before rooms were turned, the coal would buckle out. That was an extreme condition, but I wanted to emphasize that with such a deep cover there should be some limiting percentage of coal area permitted to be taken out in advance with the expectation that a large percentage of the total coal would be obtained, chiefly on the retreat.

Samuel A. Taylor.—In reply to Mr. Rice's question in regard to the practice, I will say that in the Connellsville district there is a peculiar condition existing in the coal, about halfway down a soft streak occurring in the seam. For instance, taking a section of coal seam about the middle, there is a space of about a foot which is soft, where the strength of the coal is not more than half what it is above or below. The result is that you cannot leave small pillars; they have to leave immense pillars to avoid the crushing. That is not the same physical condition as in some districts where the coal has a practically uniform hardness from the top to the bottom. Consequently, in the Connellsville district rooms are driven on 80-ft. centers, and in the Pittsburgh district on 35-ft. centers, largely, and about 25 to 28 ft. taken in the advance and a pillar left for the retreat of probably not over three or four yards. At a little mine that I had charge of, where the covering was not over 150 ft., probably not over an average of 100 ft., we used longwall machines, driving the rooms 26 ft. to 28 ft. wide and leaving a small pillar about 8 ft. thick. We did the mining of the pillars with the longwall machines, making a cut along the rib of 35 to 40 ft. We would draw one of these pillars out in about a week's time. There was no necessity of using the longest length of cutter bar; the length we used undercut 6½ ft. deep, and the coal would practically all come out without requiring the extra foot of undercutting on the gob side. When the coal was shot it would come down from the draw slate and leave only a small wedge-shaped piece of coal at the bottom in the back next the gob, and yet practically all of that coal was got out by machine mining. The coal was uniform in density from the draw slate to the bottom. There are several other districts, which are identical with the Connellsville district, where the soft binder, or soft streak in the coal, demands a very much increased pillar, not because of the overlying strata so much, as to prevent the crush by reason of requiring the rooms to remain a longer period than when operated continuously and uniformly and the coal mined out quickly. I mention that simply because you cannot be governed alone by the overlying strata of coal, but must take into consideration the uniformity of conditions.

Howard N. Eavenson, Gary, W. Va. (communication to the Secretary*).—Several members have asked the tonnage loaded per day per miner upon which the author based his figures. For the year 1913 at the mines of the United States Coal & Coke Co., for an output of

* Received June 9, 1915.

3,530,390 net tons, 14.45 tons per working day per loader was averaged, and for the year 1914, for a total output of 1,835,692 net tons, the figure was 15.52 tons per working day per loader. In addition to this, the miner cleaned his coal and gobbed the refuse, an average of at least 15 per cent. of the coal loaded, timbered the place, laid the track, and drilled and loaded his holes.

The reply to the query of E. W. Parker about the average production per man will depend upon what men are included in the number used. For all men employed, including shop, power plant, superintendents, clerks, engineers, and construction forces, the average production per man per day was, in 1913, 4.56 net tons, and in 1914, 5.49 net tons. As many of the men employed and whose names are on the roll do not work every day, the production per man working per day will average about 20 per cent. higher than this, and if the men actually working in and around the mines only are included the average production will be at least 50 per cent. more than the figure given.

For the various depths of cover encountered in our mining to date the following widths of rooms and pillars are observed, and experience has shown that these are ample.

Depth o Cover, Feet	Width of Rooms, Feet	Width of Pillars, Feet
Up to 300	18 to 24	42 to 36
300 to 500	18 to 24	57 to 51
Over 500	18 to 24	72 to 66
Up to 300	36 to 40	44 to 40
300 to 500	36 to 40	54 to 50
Over 500	36 to 40	64 to 60

To Jan. 1, 1915, the percentages of the areas covered by our 12 mines, developed and worked out, were as follows:

	Areas Developed, Per Cent.		Areas Worked Out, Per Cent.
By headings	32.4	By headings	32.4
In rooms	16.4	Chain pillars	2.9
In room pillars	28.1	Rooms	36.7
In barrier pillars	23.1	Room pillars	28.0
	100.0		100.0

For any depth of cover encountered so far in mining in the Pocahontas region, Mr. Storr's statement about the proper amount of seam to be removed in first mining is undoubtedly correct, and the sizes of pillars experience has shown to be the most economical are much larger than the sizes actually required to support the overhanging strata. The first principle of any economically planned and managed coal mine should be to drive rooms only as they are actually needed and to begin removing the pillars as soon as the rooms have reached their limits, and it is unfortunate that the limitations of output, labor supply, dirty coal, etc., too frequently prevent this being done, even when so planned.

The Limits of Mining Under Heavy Wash

BY DOUGLAS BUNTING, WILKES-BARRE, PA.

(New York Meeting, February, 1915)

THE first presentation of this paper was before the Pennsylvania Anthracite Section of the Institute in May, 1914, after which a committee was selected to verify and add to the data contained in the original paper. The author is indebted to G. W. Engel, H. W. Montz, R. V. Norris, and H. H. Otto, of this committee, for additional information. I also wish to acknowledge the courtesies extended me by a number of the mining companies for information relative to accidents, systems of proving, and mining practice. The paper should bring forth additional criticism and recommendations, which will be of value in establishing future practice relative to this subject.

Throughout the northern anthracite field of Pennsylvania glacial and alluvial deposits of sand, gravel, and clay exist in varying quantities. The extent and character of these deposits are described by N. H. Darton in *Bulletin* 45 of the U. S. Bureau of Mines, entitled, Sand Available for Filling Mine Workings in the Northern Anthracite Basin. It is estimated that the northern field contains 10,000,000,000 tons of these deposits, or a sufficient quantity to cover the 176 square miles of coal area to a depth of about 25 ft. These deposits of sand, etc., overlying the coal measures attain a maximum depth of more than 300 ft., and are usually saturated to such an extent as to render them semi-fluid. It is this condition of fluidity that limits the mining of coal seams cropping in or in close proximity to these deposits, and it is in the interest of safer and more efficient mining methods under these conditions that the study of this subject is here presented.

The dangers incident to mining under deep deposits of sand have long been realized by operators in the northern anthracite field, and much has been done to prove the contour of the rock and vein croppings underlying these water-bearing deposits of sand and gravel.

Although, as far as we know, the accidents of the past have not resulted in the loss of life, except in two instances, yet nearly all were attended with more or less miraculous escapes of men working in the vicinity of the breaks, so that the probable existence of pot holes, etc.,

brings to our attention most forcibly the necessity for exercising every care and precaution in working mines in territory of this character.

TABLE I.—*Accidents in Wyoming Field Due to Inrushes of Sand and Water*

Accident, No.	Date	Mine	Location	Vein	Tapped by	Result
1	July 4, 1872	Burroughs	Plainsville	Hillman	Breast	An inrush of sand and water A pumpman, the only man in the mine at the time, easily escaped.
2	June 30, 1874	Wanamie No. 18	Wanamie	Red Ash	Breast	Gangway and workings in the vicinity were filled for some distance from the break.
3	Jan., 1882	Maltby	Swoyersville	....	Rock Plane	Gangways were filled; also the shaft for a vertical height of 90 ft.
4	Apr. 23, 1884	Fuller	Swoyersville	Six Foot	Slope	Slope filled to the top for a distance of 900 ft.
5	1884	Ridge	Archbald	Archbald	... ..	....
6	May, 1885	Ridge	Archbald	Archbald	. .	....
7	Dec. 18, 1885	No. 1 Slope	Nanticoke	Ross	Breast	Gangways in the vicinity were completely filled in less than an hour.
8	Aug., 1889	Fuller	Swoyersville	. .. ..	Rock Plane	The plane and all workings tributary to it were filled with sand or water.
9	Mar 1, 1897	Mt. Lookout	Wyoming	Pittston	Breast	A large area of the workings was filled; no men at work at the time.
10	Dec. 30, 1898	Wanamie No. 18	Wanamie	Cooper	Breast	Gangway on lower level filled to a height of 2 ft. for a distance of 300 ft. Depression on the surface 100 ft. east and west and 75 ft. north and south.
11	Feb. 2, 1899	Franklin	Wilkes-Barre	Kidney	Breast	Filled gangway to a height of 3 or 4 ft. for a long distance.
12	Apr. 13, 1899	No. 2 slope	Nanticoke	Hillman	Breast	Gangways were filled for several thousand feet. Breasts had been worked 26 years previous; no men at work in the vicinity. Surface depression was 70 to 80 ft. deep.
13	Apr. 25, 1899	Bliss	Hanover	Hillman	Breast	Gangways and tunnel in the vicinity were filled tight to roof. Conical depression on surface 60 ft. in diameter and 40 ft. deep.
14	June 10, 1914	Sugar Notch No. 9	Sugar Notch	Kidney	Breast	Gangways and tunnels were filled tight to the roof. Depression on surface 150 ft. wide, 210 ft. long and 60 ft. deep.

To safeguard the mine workings against possible danger, some of the mining companies have formulated rules for minimum thickness

of rock to be maintained under water-bearing wash. These rules are variable. Some companies work to 100 ft. of rock over the vein when the depth of wash exceeds 100 ft., and for less than 100 ft. of wash the thickness of rock is maintained the same as the depth of wash to and including 40 ft.; others work to a uniform thickness of rock, varying from 20 to 100 ft., for all depths of wash. Then again, with the progress of mining and the more definite and closer proving of the rock contour, the minimum rock thicknesses previously established are reduced and the workings carried closer to the overlying sand deposits.

A number of accidents have occurred, due to mine workings striking into deposits of sand, etc. For the purpose of reviewing the results of these inrushes, also to compare the conditions thought to exist at the time with those subsequently proved, Table I is presented, in which the more important accidents, of which any record can be found, are given.

Accident No. 1 occurred in a breast in the Hillman vein. Its exact location does not seem to be known, but is presumably within 200 ft. of the old canal bridge. It has been stated that the cave occurred under the old canal located south of the Burroughs shaft. The canal bed over these old workings, which are also known as the Enterprise workings, ran on a course which would maintain a practically constant depth to the Hillman vein of about 68 ft. There have been no provings of the top of rock along this old canal bed in this vicinity, consequently nothing is known as to the probable thickness of rock over the vein at the point of inrush.

Some time in October, 1882, another cave occurred in the Hillman vein of the Mitchell shaft workings within 1,500 ft. of the cave which occurred on July 4, 1872. The river had risen to a considerable height so that the entire surface of the "flats" in that vicinity was covered with water. It was while this condition existed that the cave occurred. The opening on the surface was about 150 ft. long by 50 ft. wide. On Mar. 29, 1913, when the river had again risen to flood height and covered this same territory, a cave 110 ft. long and 90 ft. wide occurred in the identical spot. Again, on Mar. 30, 1914, the river having flooded this area, another cave occurred in the same location.

The section, Fig. 1, shows the old Hillman workings in the vicinity of this cave. The vein had been worked to a point very near the top of rock and the records of drill-hole provings show the rock to be very soft. It is evident that the 70 ft. of overlying wash was too heavy for the small amount of soft roof rock to withstand.

In Fig. 2 is shown a cave which occurred in 1876 in the Hillman vein, near Port Bowkley, causing a depression on the surface about 100 ft. in diameter. The vein is on a 15° pitch and was worked by chambers to a point afterward discovered to be within 15 ft. of the top of the rock. At this point a break in the rock was encountered which allowed a large

volume of water, together with sand and gravel, to enter the workings. Eventually the roof gave way across the width of the chamber, the opening afterward being found to be about 20 ft. in diameter, and a large quantity of sand, culm, and gravel was deposited in the lower gangways, filling the workings to the shaft level.

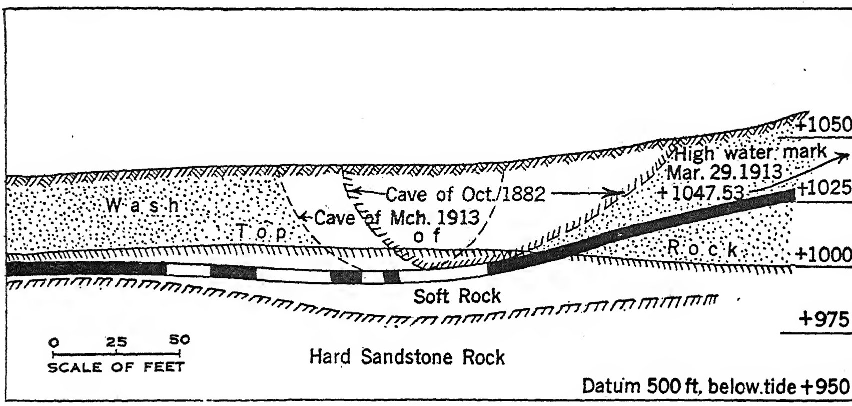


FIG. 1.—PROFILE OF HILLMAN VEIN WORKINGS.

In 1890 it was decided to remove the water and pumps were installed to do the work. When practically all the water had been pumped out of the workings, it was decided to build a dam as near as possible

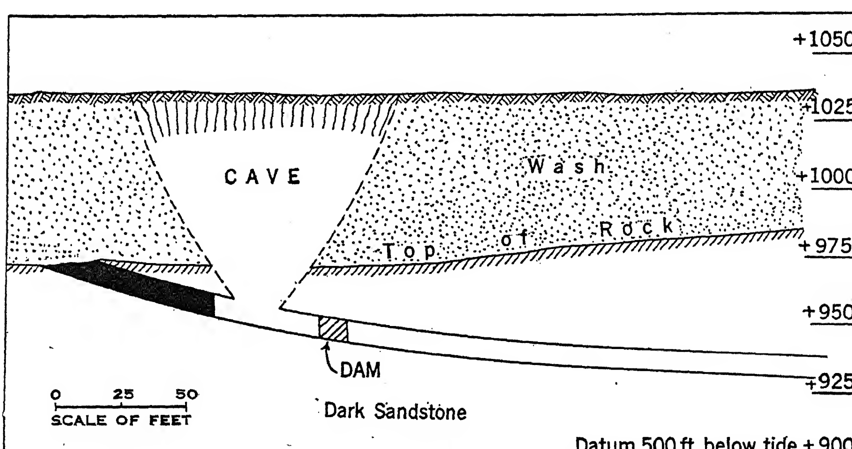


FIG. 2.—CAVE TO HILLMAN VEIN, IN 1876.

to the cave hole. In carrying out this scheme a counter gangway was driven to avoid cleaning up the gangway. It was decided to build a brick wall, as shown in Fig. 3, the ends of which were to be set in comparatively large pillars. In order to accomplish this, a number of small

pillars of coal had to be cut to allow the wall to pass through. The brick wall was 80 ft. long by 13 ft. high by 4 ft. thick. As the overlying strata of the Hillman vein were of a slaty nature, it was decided to blow down the roof in steps in order to afford the wall good support. The same was done with the bottom rock. The wall was arched to the inside from top to bottom. In order to drain the water out of the dam from time to time, it was deemed necessary to place holes in the wall. These holes were 16 in. square on the outside and 24 in. square on the inside, and were closed with wooden plugs, which were operated from the outside of the dam as found necessary.

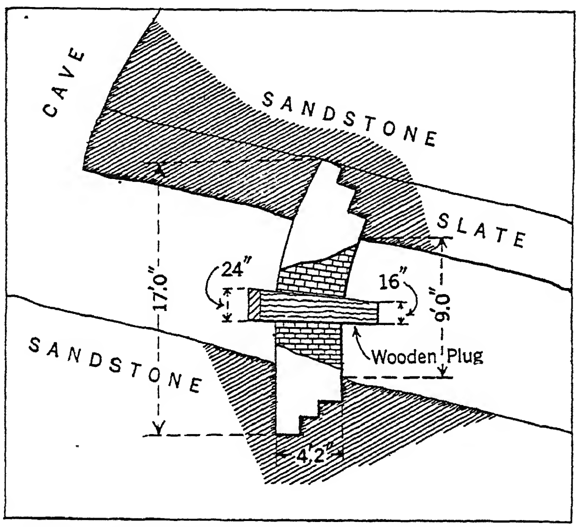


FIG. 3.—DAM IN HILLMAN VEIN, PORT BOWKLEY.

Accident No. 2 occurred in the face of a breast in the Red Ash vein. One man was killed some distance out on the gangway by being caught in the rush of sand and water. The breast in which the cave occurred was approaching an anticlinal. The face was approximately 115 ft. below the surface. Subsequent provings of the depth of sand in this vicinity indicate that an eroded channel exists on the line of the anticlinal with a width of approximately 150 ft. and a depth of 40 ft., or a depth below the surface of 65 ft. This would leave from 30 to 35 ft. of rock over the vein at the point of inrush, so it is presumed that a pot hole existed over the face of the breast at that depth.

Accident No. 3 occurred in the face of a rock plane being driven from the Eleven Foot vein to the Six Foot vein, but instead of striking the coal, it broke into sand, which in a few hours filled the mine and shaft to within about 65 ft. of the top. A drill hole 300 ft. from the break indicates approximately 14 ft. of rock over the vein, so the presump-

tion that the vein was eroded at the face of the rock plane is probably true. The depth from the surface to the top of rock is approximately 160 ft. and the depression produced on the surface was 150 ft. in diameter. A number of attempts have been made to seal the rock plane by injecting cement grout through bore holes, but without success. The elevation at which the water stands in the shaft indicates that the head of the water-bearing strata in the cave hole is approximately 65 ft. below the surface at this point. A variation in the elevation of the water in the shaft might naturally be expected to occur with any variation in the head in the water-bearing strata, which in turn could vary with the elevation of the water in the Susquehanna river.

Accident No. 4 occurred in the face of a slope immediately following the firing of a shot. The inrush of sand and water filled the slope to the shaft level, which is practically at the same elevation as the surface directly above the face of the slope. The slope has never been recovered. Drill holes in the vicinity indicate that there may have been 20 ft. of rock over the vein at the point of failure. The vertical distance from the face of the slope to the surface is 100 ft.

Accidents Nos. 5 *and* 6 were due to the presence of pot holes located about 1,000 ft. apart. In a paper by C. A. Ashburner,[1] pot hole No. 1 is said to be 38 ft. deep, 24 ft. wide by 42 ft. long on the surface and 17 ft. wide by 14 ft. long at the top of the vein. At a distance of 4½ ft. above the vein the width is given as 18 ft. and the length as 19 ft. The hole is in slate and sandy shale and the faces are extremely smooth.

Accident No. 7 occurred near the face of a counter gangway close to an anticlinal, and caused the death of 26 men, whose bodies were never recovered. On the surface above the point of inrush there was a culm bank and the depression due to the cave made a cone-shaped hole on the culm bank about 300 ft. in diameter. It has been proved by drilling close to the point of cave that the top of rock is 261 ft. below the top of the culm bank, 200 ft. being sand and 61 ft. culm. The exact thickness of rock over the vein at the point of failure is not definitely proved, but is probably about 22 ft. and positively not more than 48 ft. To the northwest of the location of this accident a considerable area was worked in the same vein without accident, although there exists but 40 ft. of rock over the vein and 174 ft. of sand and culm.

The culm bank under which this cave occurred is located on top of a hill, surface elevation + 650. Rock outcrops are found both above (+ 643 and + 594) and below (+ 567.5) the bank, and the creek valley, where deep wash might reasonably have been expected, is over 1,000 ft. south of the point of cave. The elevation of the top of the culm bank over the cave was + 697, and of the surface at the creek about + 540; the roof of workings at the point of cave, + 388. Sub-

[1] *Trans.*, xv, 636 (1887).

sequent borings in the creek valley showed a rock elevation of + 442, while the rock at the point of cave was not over + 436 and probably about + 410. Considering the rock exposure, elevation + 567, between these two, it is apparent that the buried valley split, and that besides a deep and unsuspected gorge under the hill there was probably a pot hole at the point of cave.

Accident No. 8 occurred in the face of a rock plane being driven from the foot of a slope in the Eleven Foot vein to the Six Foot vein, and at a point 1,300 ft. south of the location of accident No. 4. The plane had been driven 225 ft. on about 14° when the inrush occurred. There is no record to show that the plane struck into the Six Foot vein, but, considering the elevation to which the plane had been driven, it appears that the vein was eroded at this point, although it was thought that 40 ft. of rock existed over the vein. The depth from the surface to the face of the plane is approximately 150 ft. The inflow of sand and water filled the Eleven Foot vein workings and the shaft up to 10 ft. above the Six Foot vein, which is about the same level to which the inrush of accident No. 4 filled the Six Foot slope. The Six Foot vein in the shaft is about 15 ft. lower than the surface above the face of the rock plane, so it is evident that the water in the shaft was at the same elevation as the water-bearing strata in the vicinity of the cave.

There are reasons now to believe that an abrupt washing out of the rock was the cause of this accident as well as of accident No. 4. Within a very short distance north of these two caves there appears to be sufficient cover. The water in the Fuller mine is being lowered, and it is presumed the openings leading to the water-bearing strata are largely closed, as the water has now been lowered to a point about 16 ft. below the Six Foot landing in the shaft.

Accident No. 9 occurred in the face of a breast approaching an anticlinal, about 5 hr. after the miner had left. The place had become very wet and the water percolating through the roof was evidence of the proximity of a deep erosion, as a bore hole some distance in advance proved that 46 ft. of rock and 86 ft. of sand overlaid the vein. The breast was being driven 24 ft. wide in the Pittston vein, which has a thickness of 6 ft. Eight brick dams have been built to guard against any future starting of the sand. The building of these dams has sealed off about 13 acres of territory.

Accident No. 10 occurred in the face of a breast just started off a counter gangway. Five men were at work close by but all escaped. The vein at this location pitches 50°, and the counter gangway from which the breast was started is 95 ft. below the surface. The presence of a dangerous deposit of sand was never suspected, as the rock croppings on the surface were only 100 ft. south of the line of the gangway. Subsequent provings of the top of rock indicate that the normal depth

of sand is 55 ft., which would leave 40 ft. of rock over the gangway. The break occurred at about 15 ft. above the gangway, so there must have been a pot hole or an eroded gorge on or near the crop of the vein, with a depth of approximately 25 ft.

Accident No. 11 occurred in the face of a breast in the Kidney vein, which at this point pitches about 25°. The face of the breast became very wet and was stopped two or three days before the accident. Subsequent to the inrush a battery was built in the breast a short distance below the face and the depression on the surface filled.

In order to show how serious this accident might have been, on the night of the cave, no less than 100 people had been enjoying the skating on a pond immediately overlying this area. When the cave occurred, large pieces of ice were carried into the workings along with the mud, which filled the gangways to a depth of 3 or 4 ft. Large pieces of ice were discovered later at a distance of about 1,000 ft. from the cave. The cave occurred about 2 A.M. In the chambers east of where the cave occurred the fire clay was taken down and the "bone" propped up to afford better roof. Batteries were built near the faces of the chambers in order to withstand any possible danger of another cave.

A drill hole put down from the surface at a point 80 ft. west of the cave indicates that the maximum thickness of rock over the vein at the face of the breast in which the break occurred was 15 ft. and it is probable that it was considerably less. The depth from the surface to the point of inrush is 67 ft. On Feb. 27, 1908, a second cave occurred at the same point, presumably due to the battery, erected nine years before, giving way. This second cave produced a cone-shaped depression on the surface with a maximum diameter of 140 ft. and a depth of 40 ft.

Accident No. 12 occurred in an old breast which had been driven 26 years earlier. The face of the breast is 127 ft. below the surface and it was thought that the rock overlying the vein had a thickness of about 50 ft., but it has subsequently been proved to be not more than 20 ft. and was probably less.

This accident occurred at a point about 1,200 ft. south from accident No. 7. In Fig. 4 are shown the conditions as developed by subsequent drilling. The cave hole extended over 3½ acres of surface, and the quantity of clay, gravel, and quicksand washed into the mine approximated 400,000 cu. yd., and reached points in the workings over 4,000 ft. from the point of cave. The great amount of débris washed into the mine was largely due to the fact that the cave occurred at about the junction of Forge and Newport creeks, so that the waters of both flowed into the cave and washed in large amounts of material beyond that naturally tributary to the opening. The flow was finally stopped by throwing in large quantities of brush, prop timbers, hay, etc., and checking the flow of the two creeks by temporary dams.

Accident No. 13 occurred immediately following the firing of a blast. Men at work in the vicinity escaped.

Accident No. 14 occurred in the face of a breast being driven in the

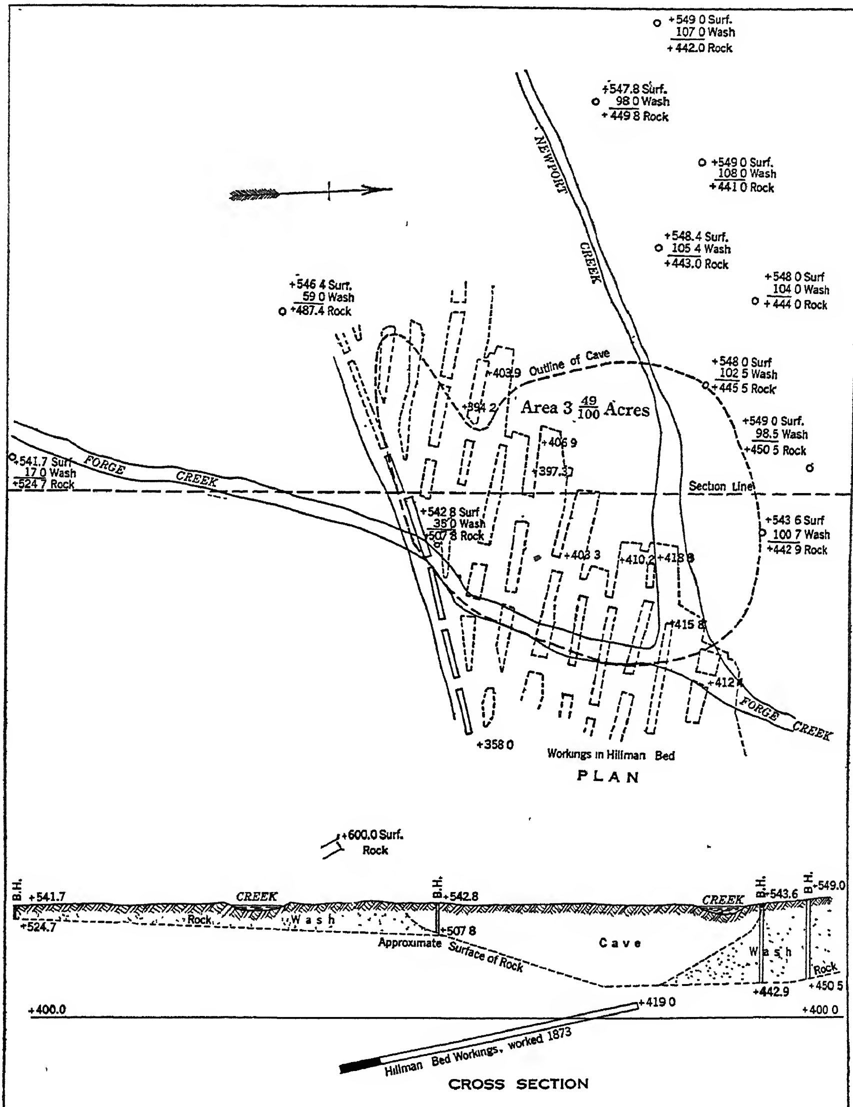


FIG. 4.—PLAN AND CROSS-SECTION SHOWING LOCATION OF ACCIDENT NO. 12, APRIL 13, 1899.

Kidney vein, immediately following the firing of a blast. Men working in the vicinity were immediately notified; those working on the inside of the cave escaped through a hole in the rock leading from the Kidney

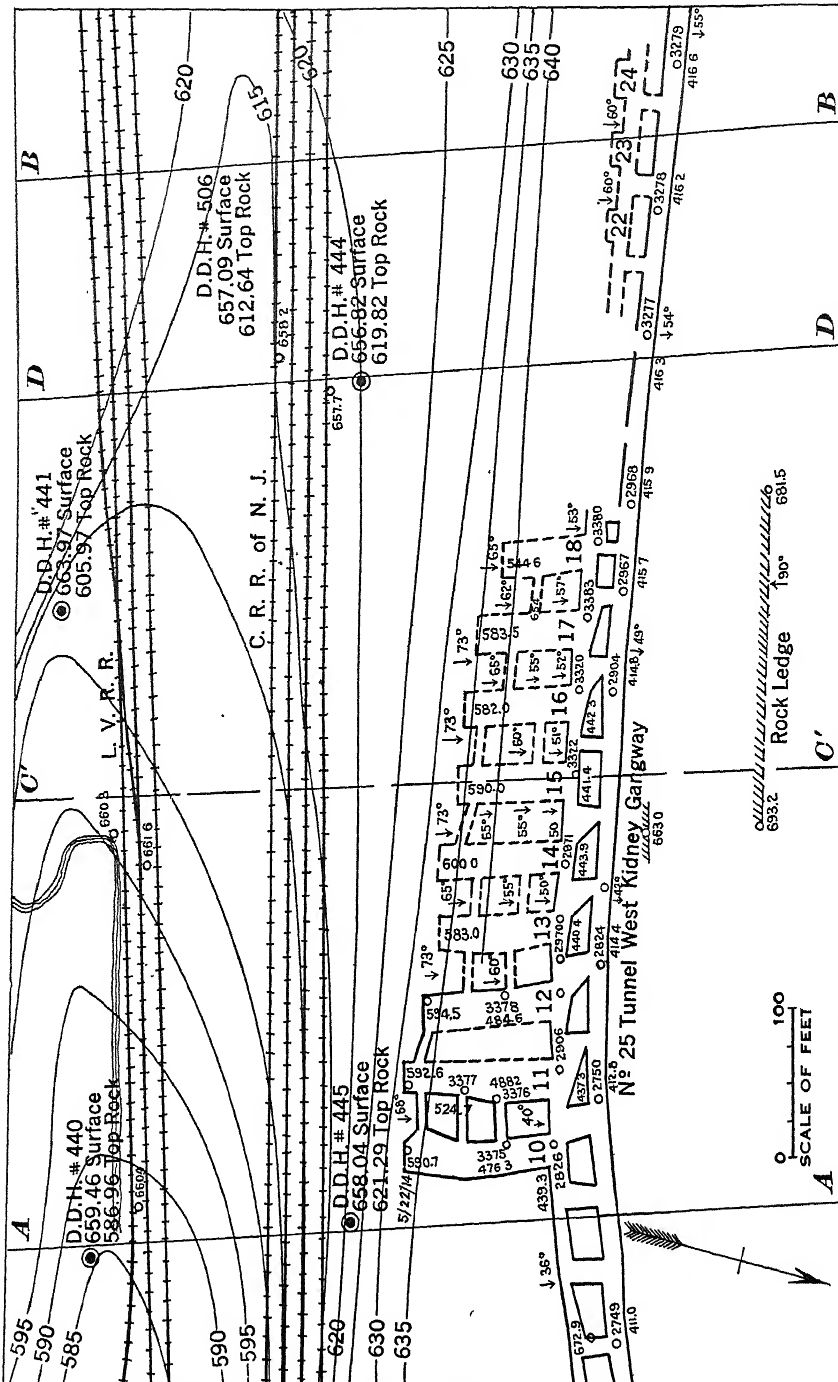


Fig. 5.—Mine Workings and Surface Features Previous to Accident No. 14.

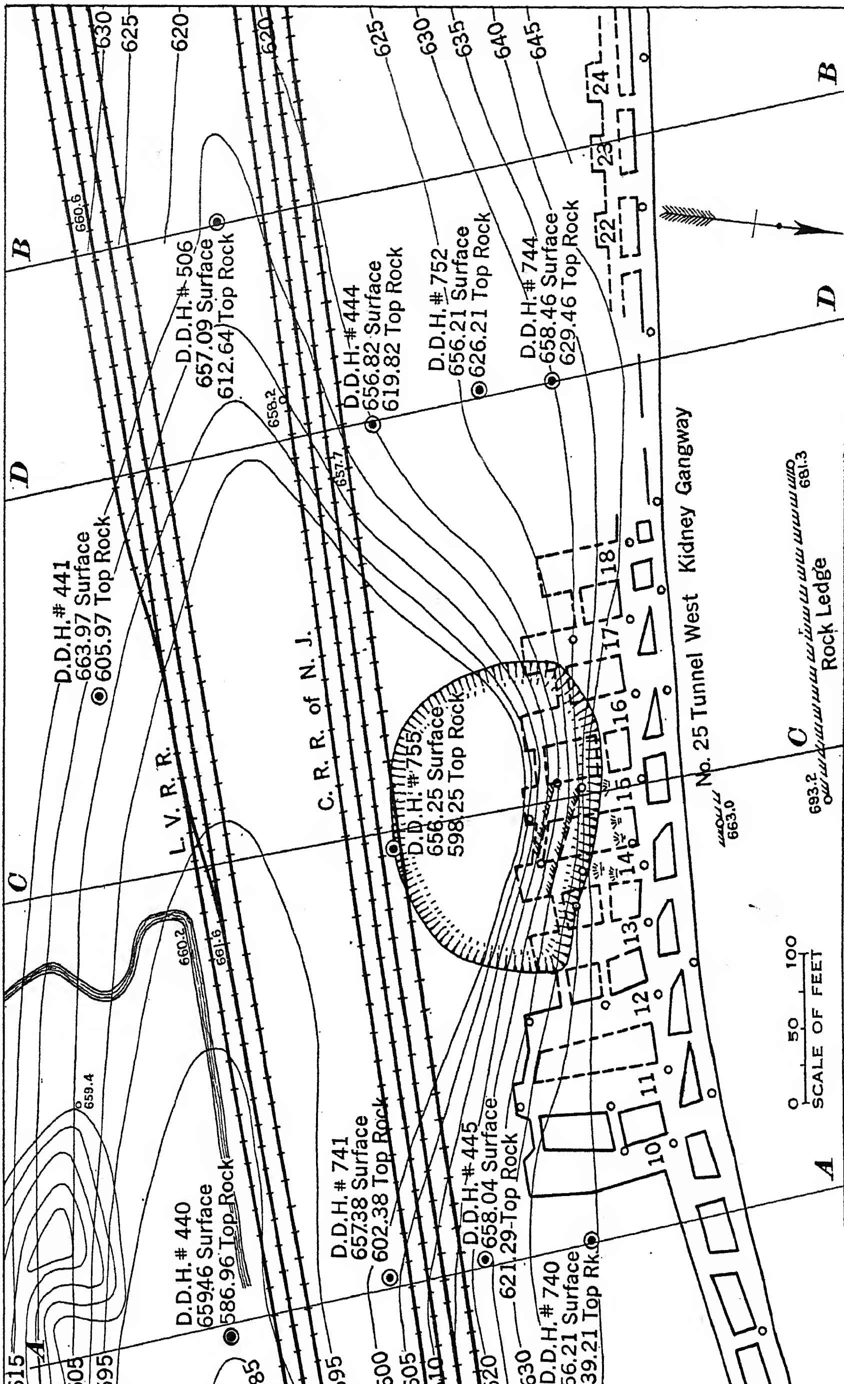


FIG. 6.—SURFACE CONDITIONS SUBSEQUENT TO ACCIDENT NO. 14.

vein to the surface at a considerable distance inside of the cave, and those working on the outside of the cave escaped to the hoisting shaft. The cave occurred about 11 o'clock in the morning, the first evidence on the surface being a conical hole which continued to increase in size for 5 or 6 hr. The material carried into the mine consisted principally of sand and clay in a semi-fluid state, from a swamp in which the cave occurred.

Approximately 20,000 cu. yd. of solid material was carried into the mine, which filled or partly filled several thousand feet of gangways and tunnels and required months to clean up.

In Fig. 5 are shown the mine workings, contours of top of rock, location of drill holes, and the principal surface features as they existed

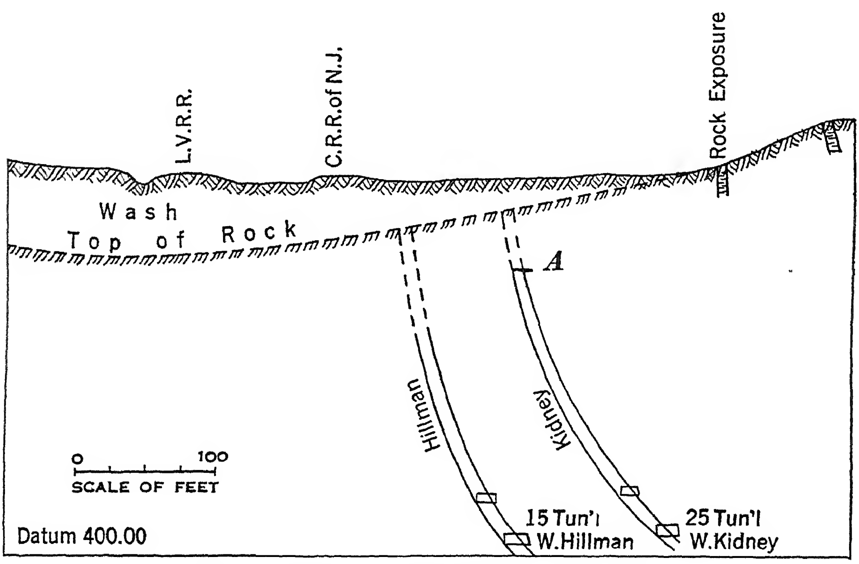


FIG. 7.—SECTION ON LINE C′–C′ OF FIG. 5.

previous to the accident. The cave occurred at the face of breast 15, where the vein pitches 75°.

In Fig. 6 are shown the conditions after the cave. It is to be observed that additional drill holes were put down and more accurate contours of the top of rock secured.

In Fig. 5 and Fig. 7, section C′–C′ illustrates the conditions as they were considered to exist before the accident. The face of breast 15 at the time of the cave was at *A*, or at an elevation of + 590.0. It was intended to drive this breast to an elevation of + 600.0, as the elevation of the top of rock immediately over this breast was thought to be + 630.0. The elevation of the surface directly above is + 657.0.

In Fig. 8, section C–C illustrates the conditions after the accident,

which conditions are also shown in plan in Fig. 6. The top of rock at the point of cave is at elevation + 600.0, which is only 10 ft. above the face of the breast at the time of the accident.

The difference in the contour of the top of rock on these two sections is very striking, still the top of rock on sections approximately 300 ft. east and west of section C–C has not been materially changed by drill-hole provings made subsequent to the time of the accident.

At the Exeter colliery, near West Pittston, in proving the top of rock in advance of mining in the Pittston vein, a pot or gorge was developed. This erosion has a maximum depth of 110 ft. and a width on top of about 200 ft. The depth from the surface to the bottom of the pot or gorge is 127 ft. Two holes 100 ft. apart and at right angles

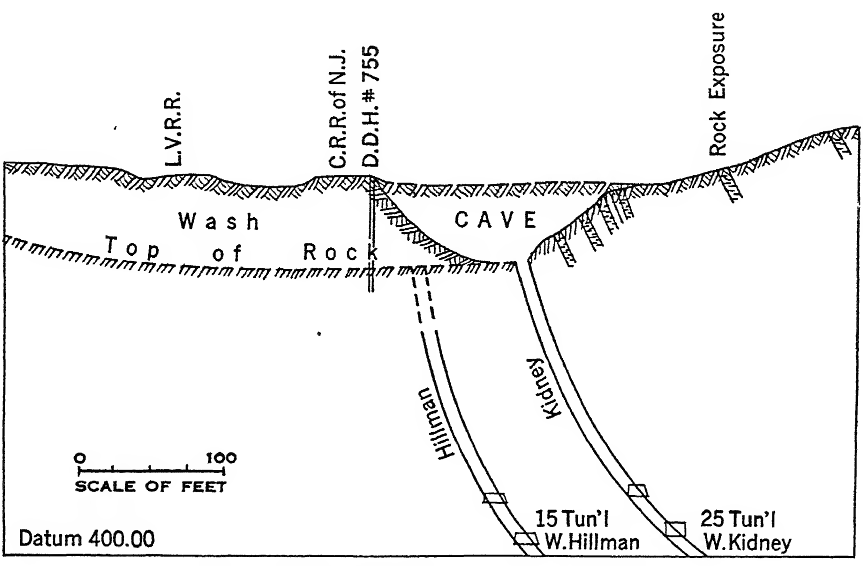


FIG. 8.—SECTION ON LINE C–C OF FIG. 6.

to the holes proving a width of 200 ft. show the same depth of erosion, so it appears that it may be an eroded gorge, but if a pot, it is probably elliptical in shape. An indication that the erosion is a gorge is given by considering the rock measures. The top of rock immediately north of the erosion is apparently the bottom rock of the Checker vein and at the location of the deep erosion there is apparently a broken anticlinal, which would give cause for the deeper erosion. The proving of top of rock in this locality was being done with holes spaced 200 ft. centers when the deep erosion was discovered.

A very interesting proving of rock erosion at the Stearns colliery, near Nanticoke, indicates the presence of a deep gorge with a width on top of about 100 ft. and a depth on one side of 95 ft. and on the other

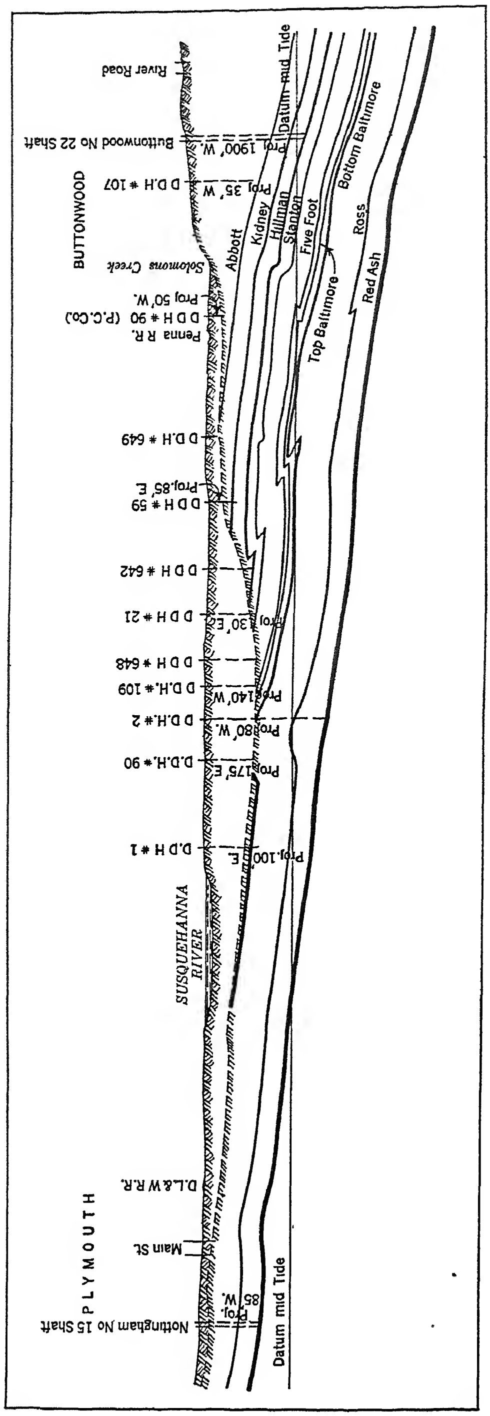


FIG. 9.—CROSS-SECTION OF THE WYOMING VALLEY.

side of 45 ft. The bottom of the gorge is 105 ft. below the surface. The gorge as developed indicates, in consideration of provings of the top of rock on sections to the east and west, that it is a contraction in the eroded channel, and like the so-called pot hole over the Pittston vein at the Exeter colliery, is probably nothing more than an eroded gorge. These possible conditions exemplify the importance of contouring the top of rock in provings of rock erosions.

In Fig. 9 is shown a cross-section of the Wyoming valley from Plymouth to Buttonwood, on a line west of the Nottingham and Buttonwood shafts. This section is through the deepest point proved in the buried valley, where a depth of 309 ft. of sand, etc., was proved. The territory proved in this part of the buried valley is approximately 2 miles long and 1 mile wide and to date has required the drilling of more than 200 holes. The holes are spaced approximately 500 ft. apart and the various veins approaching the sand deposit have been worked to a uniform rock cover limit of 100 ft.

In proving the top of rock by drilling, some of the mining companies have pursued definite methods of hole spacing. Some companies space the holes on 200-ft. squares, except where the depth of sand is comparatively shallow, when the holes are spaced on 100-ft. squares. Others have spaced the holes approximately 200 ft. apart on the drilling section lines, which section lines are parallel and approximately 500 ft. apart. Additional holes are drilled intermediate to the section lines to prove any probable deeper wash. Other companies have followed no definite rule as to hole spacing. The depth to which holes are drilled into the rock is almost universally 10 ft., except where holes are drilled deeper for the purpose of proving the location and character of some underlying coal vein.

It is advisable to have all measurements of drill holes and the cores checked by a representative of the mining company, also the drilling of at least 5 ft. in the rock should be witnessed by the same representative before finally measuring the hole.

In developing the top of rock under sand deposits by drilling, it is advisable to construct sections on the lines of drilling and also contour maps of the top of rock. Such sections and contour maps are of great value in proving the possible existence of gorges or abrupt erosions, and at the same time are of further value in determining the most desirable locations of drill holes and the lines of working limits for definite rock cover.

Numerous tests of various stones have proved that sandstones take permanent sets for the smallest loads, whereas granite and limestones are nearly perfectly elastic. It has also been proved by tests on various stones that the modulus of elasticity in compression is practically the same as in cross bending, but no fixed relation has been deter-

mined of the compressive, tensile, or shearing strength of the various kinds of stone.

The shearing strength of sandstones and slates per square inch is generally slightly in excess of the modulus of rupture, and the compressive strength of various stones is variable and of comparatively little consequence here, as the compressive strength of even the lightest sandstones ranges from 4,000 to 6,000 lb. per square inch.

The moduli of rupture of various kinds of stone as given by a number of authorities are shown in Table II.

Table II.—*Moduli of Rupture of Stones*

	Maximum	Minimum	Average	Authority
Blue stone flagging	4,511	360	2,700	Baker.
Slate	9,000	1,800	5,400	Baker.
Slate	11,230	7,425	....	Arsenal tests, 1902
Slate	....	....	8,480	Merriman.
Granite	2,700	900	1,800	Baker.
Granite	....	....	1,754	Merrill.
Granite	....	....	2,610	Arsenal tests, 1907.
Granite	....	....	1,667	Arsenal tests, 1905.
Granite	....	....	1,365	Bauschinger.
Granite	....	....	1,194	Bauschinger.
Glass	3,500	....	....	Church.
Glass	4,132	....	....	Fairbairn.
Sandstone	....	....	1,576	*Technology Quarterly.*
Sandstone	....	....	1,200	Merriman.
Sandstone	1,273	655	....	Arsenal tests, 1895.
Sandstone	2,243	1,500	....	Arsenal tests, 1895.
Sandstone	2,340	576	1,260	Baker.
Sandstone (variegated)	....	....	469	Bauschinger.
Sandstone (variegated)	....	....	718	Bauschinger.
Sandstone (variegated)	....	....	1,109	Bauschinger.
Sandstone (variegated)	....	....	341	Bauschinger.
Sandstone (Carboniferous)	....	....	483	Bauschinger.
Sandstone (slaty)	....	....	249	Bauschinger.
Sandstone (slaty)	....	....	135	Bauschinger.
Sandstone (green)	....	....	156	Bauschinger.
Sandstone (Cretaceous)	....	....	597	Bauschinger.
Sandstone (Cretaceous)	....	....	967	Bauschinger.
Gray stone	2,200	....	....	Kent.
Light stone	1,170	....	....	Kent.
Stone	....	....	2,000	Merriman.
Quartz conglomerate	....	....	654	Merriman.

From the results of tests as given in Table II, the average moduli of rupture for the various stones can be considered as follows:

	Pounds per Square Inch
Blue stone flagging	2,700
Slate	7,736
Granite	1,681
Glass (maximum)	3,866
Sandstone	806

Safe unit stresses for various stones have been given by many authorities and I give in Table III the stresses in pounds per square inch recommended by W. J. Douglas as illustrative of possibly a fair average of such values.

TABLE III.—*Safe Unit Stresses for Stone*

	Compression Pounds per Square Inch	Shear Pounds per Square Inch	Tension Pounds per Square Inch
Blue stone flagging	1,500	....	200
Granite	1,200	200	150
Limestone	800	150	125
Sandstone	700	150	75

It is to be observed, in the case of sandstone, that a safe tensile strength of 75 lb. and a shearing strength of 150 lb. per square inch are given. Now, in consideration of the fact that the modulus of rupture is invariably in excess of the tensile strength, also that the resistance to shear slightly exceeds the modulus of rupture, a value of 100 lb. per square inch for the modulus of rupture of sandstone would be consistent. This assumed value of 100 lb. per square inch as the modulus of rupture for sandstone beams compared with the average modulus as derived from Table II gives a safety factor which, although slightly less than the factors generally used for stone in building construction, seems to be sufficient for the conditions of this problem.

When sandstones and slates, which generally overlie the coal veins, are considered as beams or slabs spanning mine openings for the support of overlying strata or other superimposed load, their transverse strength is of first importance. The ability of such material to serve as a beam depends upon its tensile strength, since that is always less than its compressive strength.

In the formula for flexure $\frac{pI}{e} = M_m$, we can compute the value of p, the greatest normal stress in any outer element, when all other quantities are known. This value of p is theoretically correct provided the elastic limit of the material has not been exceeded. This formula has also been used for the rupture of beams by flexure by calling the value of p thus obtained the modulus of rupture, R. The value of R may be found to differ considerably from the unit tensile strength of some materials and is frequently much larger, having a value between the

ultimate tensile and the ultimate compressive strength. This might be expected, since, even supposing the relative extension of the fibers to be proportional to their distances from the neutral axis as the load increases toward rupture, the corresponding stresses, not being proportional to these strains beyond the elastic limit, no longer vary directly as the distances from the neutral axis; and the neutral axis does not necessarily pass through the center of gravity of the section. It is evident that the modulus of rupture does not express the actual stress in the extreme fiber of the beam, but is a quantity useful only as a basis of comparison.

The manner of failure under test depends upon the kind of material. Stones and brittle materials fail by the fibers breaking in tension, whereas, timber and the more plastic materials fail by crushing. Stone and plain concrete beams of deep section fail sometimes by shearing on a diagonal plane near the point of support.

In deriving a formula for computing the breaking load of a slab of stone from the formula $\frac{pI}{e} = M_m$, let W represent the distributed loading plus the weight of the beam itself in pounds; let b, d, and L represent the breadth, depth, and span, respectively, in inches; let R equal the modulus of rupture, in pounds per square inch.

The maximum bending moment for a constrained or prismatic beam is equal to $\frac{WL}{12}$. By substituting in the formula for flexure $\left(\frac{pI}{e} = M_m\right)$ we obtain the formula $W = \frac{2bd^2}{L} R$. Likewise, the maximum moment at the center of such a beam being equal to $\frac{WL}{24}$, the formula becomes $W = \frac{4bd^2}{L} R$.

So it is evident that failure of flexure would theoretically take place at the points of support and not at the center of the span.

In applying the formula $W = \frac{2bd^2}{L} R$ to the case of a slab spanning a breast or other mine opening, the weight of the overlying material will be taken at 108 lb. per cubic foot, and the depth of the opening below the surface will be designated by d′ in feet.

We will then have $W = \frac{108\,Ld'}{12}$, which would be the loading of the slab with a breadth of 1 ft. Substituting this value of W in the equation $W = \frac{2bd^2}{L} R$ and simplifying, we obtain:

$$d^2R = \tfrac{3}{8}L^2d'.$$

By taking the width of the opening, L, as 24 ft. and 100 lb. per square inch as the value of R, the formula becomes $d = 17.63\sqrt{d'}$,

which becomes $d = 1.47 \sqrt{d'}$ for values of d in feet. For 120 lb. per cubic foot this becomes $d = 1.55 \sqrt{d'}$, so, for convenience in use and without appreciable error, this is taken as $d = 1.50 \sqrt{d'}$.

From this formula Table IV has been prepared, which gives the thicknesses of rock for depths ranging from 10 to 300 ft. and 24 ft. width of chambers.

In the use of the formula derived for determining the minimum safe thickness of rock over mine openings for various depths below the surface, consideration must be given to a number of conditions, the more important of which are:

1. Nature of the top immediately above the coal seam, its comparative strength and liability to disintegration upon exposure to the atmosphere.

To compensate for this condition, the allowance to be made can be determined by observing the nature of the top in places that have been mined nearest to the location under consideration. Every vein and section may present different conditions, so that allowances varying from 1 to 10 ft. may be found necessary.

2. Nature and thickness of the vein, the ability of the pillars to resist squeezing, and the liability of disturbance to the overlying strata, due to caving or squeezing in underlying veins.

To compensate for these conditions is more difficult, for the reason that a squeeze will crack the overlying strata and admit water in varying quantities, although the thickness of rock cover may be far in excess of any requirement against caving. Consequently, there would be no necessity for making any allowances in the rock cover thickness for these conditions. It would, however, be advisable to compensate against squeezing in the vein, over which the rock cover is being established, by mining with a larger factor of safety.

3. Probable errors in relative vertical location of top of rock and mine workings.

To compensate for possible errors in vertical locations an allowance of 5 ft. in the thickness of the rock should be made. The compensation for this condition will depend upon the verification of surveys on the surface with those in the mine, and unless the elevations are absolutely verified an allowance of at least 5 ft. in the rock cover should be made.

4. Possibility of the existence of deep gorges and pot holes.

To compensate for possible gorges and pot holes an allowance of 35 ft. should be made.

The thickness of rock cover required for 24 ft. width of chambers to support safely the overlying strata at various depths is graphically illustrated in Fig. 10.

The curve *A* is derived from the equation $d = 1.5 \sqrt{d'}$ and repre-

sents the minimum thicknesses of rock where all conditions are absolutely known. This curve could rarely, if ever, be considered in practice, for the reasons before given.

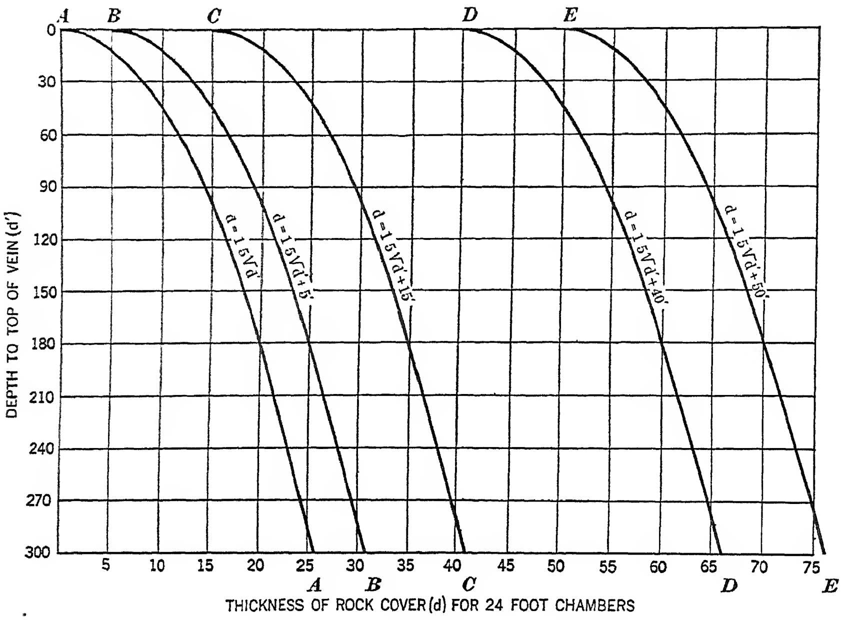


FIG. 10.

TABLE IV.—*Minimum Thickness of Rock for Various Depths below the Surface, over Chambers 24 Ft. Wide*

Depth d', Feet	Thickness of Rock d, Feet	Depth d', Feet	Thickness of Rock d, Feet
10	4.74	160	18.97
20	6.71	170	19.55
30	8.21	180	20.13
40	9.48	190	20.67
50	10.61	200	21.21
60	11.62	210	21.74
70	12.55	220	22.25
80	13.41	230	22.75
90	14.23	240	23.23
100	15.00	250	23.72
110	15.73	260	24.18
120	16.43	270	24.65
130	17.10	280	25 10
140	17.75	290	25.54
150	18.37	300	25.98

Curve *B* represents the minimum values for the thickness of rock cover *d* after making an allowance of 5 ft. for probable errors in relative elevations, and would only be used when the top rock of the vein is hard, strong sandstone or other rock of equivalent strength and durability. This curve could only be considered when the contour of the top of rock is absolutely known and there is no possibility of the existence of pots or deep eroded gorges.

Curve *C* represents the minimum values for the thickness of rock cover after making an allowance of 10 ft. for poor top in addition to the 5 ft. for probable errors. This curve could only be considered when other conditions as required for curve *B* are known.

Curves *D* and *E* represent the values for the thickness of rock cover after making an allowance in each case of 35 ft. for pots and gorges in addition to the allowances made for curves *B* and *C* respectively. Curve *D* would be used when the top is hard sandstone, and curve *E* when the top is soft and disintegrating.

Conditions may warrant the reduction of the allowances as used in determining curves *D* and *E*, but unless these conditions are positively proved, it would not seem advisable to mine with less rock cover than is represented by curve *D*.

The practices in mining under wash of various depths have not been very consistent, and it is with the idea of standardizing future practice and at the same time securing more safety and efficiency in mining under these conditions that this paper is presented.

Discussion

Arthur Hovey Storrs, Scranton, Pa.—I know something about accidents Nos. 4 and 8. At the Fuller mine there was no disturbance of the surface in connection with either of those accidents, there was no surface settling, nor signs of sand in the mine, and seemingly the water all came from a subterranean basin or water-bearing stratum overlying the Six Foot vein.

A similar thing was shown, in an accident not referred to in Mr. Bunting's table, at the Avondale, where we had an inflow of water, as high as 12,000 gal. a minute, for a considerable period.

Regarding the second accident (No. 8) at the Fuller mine: after the first inflow of water (No. 4 of the table), the mining was abandoned in the Six Foot vein. A long slope was driven in an underlying seam, and when this had reached a point about 2,000 ft. southeast of the place where the first inflow of water occurred, and where the bore holes show there was apparently sufficient rock covering for the Six Foot vein, a rock plane was driven up to the upper vein (Six Foot) and one afternoon this exposed the coal. During the following night the inflow of

the water occurred. There were absolutely no workings at this point so that no disturbance of the overlying rocks could have occurred.

The result of the inflow of water was to fill up the entire lower-vein workings in about two or three days and caused the abandonment of the mine.

After several years, considerable time and money were spent in an effort to hoist the water from the shaft and mine workings, with varying, but in the main rather indifferent, success until, due to the breaking of one hoisting rope, the engineer pulled the other hoisting tank up through the tower and down on to the engine, making a complete wreck of the whole plant. It was then decided to allow the mine to soak and it is still doing so, other portions of the property being mined from adjoining collieries.

Later came the big inflow that I spoke of, the 12,000 gal. per minute at the Avondale. In this case there was much disturbance of the surface; the depression showing from 4 to 6 ft. over an area of several acres. The surface covering varied from 6 to 80 ft. of gravel, sand, and clay, with the Susquehanna River flowing over same not far distant. It was generally contended that the water came from the river through these surface materials, but soon after the inflow a rise of the river put about 6 ft. of water over the sunken area without showing any increase of the inflow into the mine. The heavy original rate of inflow continued for some weeks and then gradually decreased until it reached a minimum of about 4,000 gal. a minute, at which rate it would flow for months. Then after a long rainy period the inflow of water would gradually increase in volume. Apparently there was a subterranean basin of water which, standing full and being tapped by the mine caving, caused the original heavy rate of inflow. This gradually drained down until the minimum inflow represented the ordinary normal amount of ground water made in this basin. Heavy continued rains gradually increasing the amount of ground water, a higher rate of inflow would be maintained for a period. The amount of water over, or in, the immediately overlying surface gravel and sand and the height of the nearby river did not apparently affect the rate of inflow.

Similar conditions to those of this Avondale inflow apparently have existed at other places in the Wyoming Valley where inflows of water have occurred. It would appear that perhaps there was a large subterranean water stratum from which these water inflows came, rather than from the surface gravel and sand. The analysis of the inflowing water at the Avondale was markedly different from that of the river water.

Charles Enzian, Wilkes-Barre, Pa.—I desire to offer a suggestion in connection with accident No. 3, which occurred in a mine adjoining the one Mr. Storrs has spoken of. The mining engineers of the Wyoming

Valley have for many years suspected the existence of a subterranean water channel in the vicinity under discussion. I think substantial proof of its existence may be taken from the experience in the construction of domestic water-supply wells. In the bottom of the upper water-bearing strata there is a clay bed of variable thickness. If the wells are sunk only to the clay they will retain the water and yield a good supply, but if the clay bed is pierced the water is lost.

Following the occurrence of accident No. 3, and directly over it, an inverted cone-shaped surface depression existed, measuring about 175 ft. in diameter at the surface and said to have been originally from 20 to 30 ft. deep. It is possible that in the accident to which Mr. Storrs refers, the water in the sub-channel was tapped in such manner as not to draw in the entire sand strata; whereas the break in the No. 3 accident apparently was large enough to draw in the clay bed and extend to the surface.

Douglas Bunting (communication to the Secretary*).—The suggestion that tests be made of the different kinds of stone overlying the various coal seams is a good one. In the making of such tests, which would be primarily for transverse strength, it would be advisable also to determine their compressive strengths for various ratios of height to least lateral dimension of specimens.

With reference to the Illinois limestone mine, with longwall coal mine underlying, and the instance where there was an inflow of 12,000 gal. of water per minute into an anthracite mine: In these instances it is probable that the failures were due to insufficient or improper distribution of pillar support, and that being the case, the discussion of these cases would not probably come under the subject of this paper, for reasons given on p. 195 in the second condition for consideration of the use of the formula for determining minimum safe rock thickness.

Relative to accidents Nos. 4 and 8 at the Fuller mine: There is no evidence of disturbance upon the surface now, and there may not have been at the time of the accidents. That being the case, comparatively little sand could have entered the mine, although it has been stated at various times that the inrushes consisted of sand and water. It would, however, be natural to assume that accident No. 8 was accompanied with an inflow of more sand than accident No. 4 and, consequently, more cause for surface disturbance.

The water in the Fuller mine is being removed and possibly, at some future time, we will know something more definite as to the quantity of sand in the mine at the location of these two accidents; also with the lowering of the water it will be proved to what extent the openings which caused these accidents have been closed and what effect the years of standing have had upon their permeability.

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Recent Developments in Coal Briquetting

BY CHARLES T. MALCOLMSON, CHICAGO, ILL.

(New York Meeting, February, 1915)

In the United States, improvements in methods of combustion have made possible the use of the smaller sizes of anthracite. This coal is now being reclaimed from the culm banks accumulated by the miners in the more prodigal years. To-day the freshly mined "slush" is segregated and stored against the time when, through briquetting or some other means, it may be utilized as a commercial fuel.

For many years Europe has achieved enviable results in the conservation of her small coal by briquetting. This profitable industry, first placed on a commercial basis in France in 1842, has grown to such magnitude that the production during 1913 exceeded 36,000,000 short tons. Briquets produced in Great Britain are used principally for steam purposes and the greater part of the output is either exported or taken by the Admiralty. On the Continent the briquets are sold in all branches of the coal trade. The French railways consume from 16 to 40 per cent. of their fuel in briquet form, and in Germany the lignite briquets (of which 20,000,000 tons were produced in 1913) are as familiar and interesting to the American traveler as the huge porcelain stoves in which they are burned.

It is not strange, therefore, that aggressive and far-sighted promoters heralded this means of converting our ubiquitous coal waste into revenue. To the superficial student the briquetting process seemed so simple as to make success easy. It is for this reason principally that the path of the industry in America has been strewn with failures. The rash enthusiast had either not worked out the mechanical problems involved or had failed to grasp the local conditions under which the briquets must be made and sold. It may be confidently stated, however, that in spite of the many vicissitudes through which it has passed, coal briquetting is now an accomplished economic success in this country, and as a profitable adjunct to the coal industry it has come to stay.

Several valuable contributions relating to coal briquetting are found in the *Transactions*. Charles Dorrance, Jr., has sketched briefly the history of the anthracite briquetting plants from the time of Loiseau's pioneer plant down to that of the Lehigh Coal & Navigation Co. at Lansford. Since then this phase of the industry has remained practically at

a standstill in the East; in fact, several of the plants then in operation have ceased to exist.

The experience of the past few years points to the conclusion that the anthracite product as now manufactured is not what the consumer wants. Briquets marketed on the Eastern seaboard are nearly all made with coal-tar pitch as a binder, following the European practice, and are sold for domestic purposes, at from $1 to $1.50 below the price of chestnut coal. While the culm may not contain more ash than is allowed by the government specification for prepared sizes of anthracite, the ash is thoroughly distributed through the briquet, and this feature, combined with conditions under which the fuel is consumed, causes the binder to distill off at lower temperatures than the coal burns, thus producing some soot and smoke. While this defect is not great, the difference in price does not seem to compensate for it. Other binders which will eliminate this objectionable feature are now available.

There is, however, an indifference on the part of the large anthracite operators which is not encouraging. It has been stated that every means is being employed to increase the production of anthracite, and that the demand is considerably in advance of the supply, and is growing steadily. However, the fact that coal is being stored at the present time does not bear out these statements. As one operator puts it, so long as there is a surplus of the high-priced sizes, no effort will be made to deplete the market for these sizes by the introduction of briquets from the low-priced sizes.

On the other hand, it is encouraging to observe that the plant of the Scranton Anthracite Briquette Co. at Dickson City, Pa., is selling its entire output of 300 tons per day for locomotive consumption, and that these pitch-made briquets produce no deleterious effect. The briquets are mixed with bituminous coal and are proving a successful engine fuel.

It is not strange that one of the earliest practical demonstrations of the briquetting industry took place on the Pacific Coast. Since California produced only inferior grades of lignite, high-priced Welsh bituminous and anthracite coals, Australian and some Japanese coal were imported in considerable quantities up to the time that the better grades of native coals from the Rocky Mountain States and Washington entered this market. Robert Schorr has already presented to this Institute a description of the plants built by him at Stockton and Oakland. The latter plant is still intact, but the supply of screenings produced in the handling of imported coals is now so small as to render briquetting no longer profitable.

In the meanwhile the most important progress in the briquetting of bituminous coals has been made in the Middle Western States. I will outline briefly the plants which mark this development.

The greatest single impetus to the industry was given by the U. S.

Geological Survey through its experiments at St. Louis and Norfolk from 1904 to 1907. During this period the Semet-Solvay Co. erected a plant at Detroit to utilize its accumulations of coke breeze. Results at this plant demonstrated the necessity of equipment designed to meet American conditions. The subject has been well covered in a paper by W. H. Blauvelt before this Institute.

As a direct result of the government work, the Western Coalette Fuel Co. built a plant at Kansas City, Mo., using an 8 ton per hour Renfrow press of the plunger type producing 12-oz. briquets. This plant was unsuccessful because the mechanical problems had not been given proper consideration.

Profiting by this experience, the Renfrow company built another press having a capacity of 8 to 9 tons per hour, and making briquets weighing 13 oz. This press was installed in the Norfolk plant of the Survey and made the briquets which were tested on the Eastern railroads and for the Navy Department. Upon concluding the Norfolk tests the machine was sold to the Rock Island Coal Mining Co. and installed at Hartshorne, Okla.

The Hartshorne plant operated about three years on a successful commercial basis, briquetting the bituminous slack mined by the company. The product was marketed under the trade name of "Carbonets." When this project was undertaken, slack coal was accumulating rapidly, as the Mexican market for McAlester coke had ceased to exist, and the coke ovens which had consumed this slack were shut down. Later, however, a prolonged strike in this region, together with the decrease in Texas oil production, and the opening of new markets in the Southwest, soon made the price of slack prohibitive for briquetting.

It must be borne in mind that the most important commercial feature in the marketing of briquets is the margin between the selling price of the fine coal to be briquetted, and that of the prepared coal which the briquets meet in competition. It is principally because of the variableness of this margin that the coal-briquetting industry has not been successful in every part of the United States where it has been tried.

The Hartshorne plant demonstrated the fact that the briquets could be sold when a uniform product was manufactured. Briquets weighing from 8 to 16 oz. met the market requirements in the Middle West as a competitor for 3-in. bituminous or anthracite coal. Simpler and more efficient machinery, however, was required. In 1909, the Standard Briquette Fuel Co. built a plant at Kansas City, Mo., to try out a 10 ton per hour plunger press, following the German and English design but modified to meet American conditions as to size of briquets. The plant was an improvement over former designs, but after two years of costly experimentation the press was replaced by another of the same tonnage. Even after the second press was perfected mechanically, during a period of

two years, its operating cost was too great to make briquetting profitable on the margin obtaining at Kansas City.

About this time a plant of somewhat similar design and using a Renfrow press was built by the Detroit Coalette Fuel Co. It is still in operation, producing 12-oz. briquets out of Pocahontas coal at the rate of 8 tons per hour.

The same necessity for conservation which found a use for the hitherto waste coal in the culm banks of eastern Pennsylvania, found a market for the fine coal in the great Mississippi Valley region. This market was created by the development of the mechanical stoker, and as a result the prices of slack and lump coal were more nearly equalized. Hence it transpired that the future of bituminous briquets must be limited to domestic use, and that the industry could be made profitable only by developing highly efficient machinery of large tonnage. The railroads

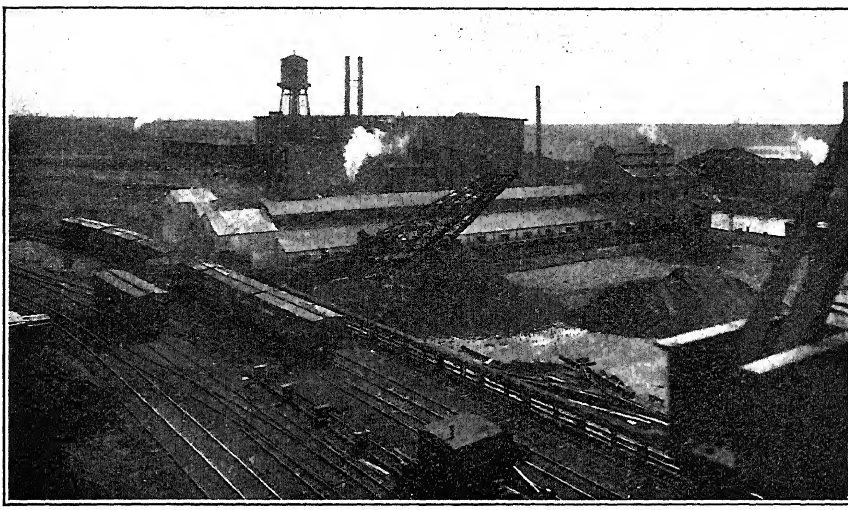


FIG. 1.—BRIQUETTING PLANT OF BERWIND FUEL CO., SUPERIOR, WIS.

and large coal operators could be interested only on this basis. The Rutledge press was designed to meet these conditions.

The St. Louis Briquette Machine Co. began in 1909 the construction of a plant at Livingston, Ill., to be used in perfecting the Rutledge press, and to learn something of the value of Illinois briquets. The plant had a capacity of 32 tons per hour of 16-oz. briquets. In 1911 the press had been sufficiently developed to warrant the Berwind Fuel Co. in contracting for the construction of a plant on its dock at Superior, Wis. (Fig. 1), which was completed the following year.

A description of the Berwind plant has been published in the technical press, and need not be repeated here; however, the reasons for establishing

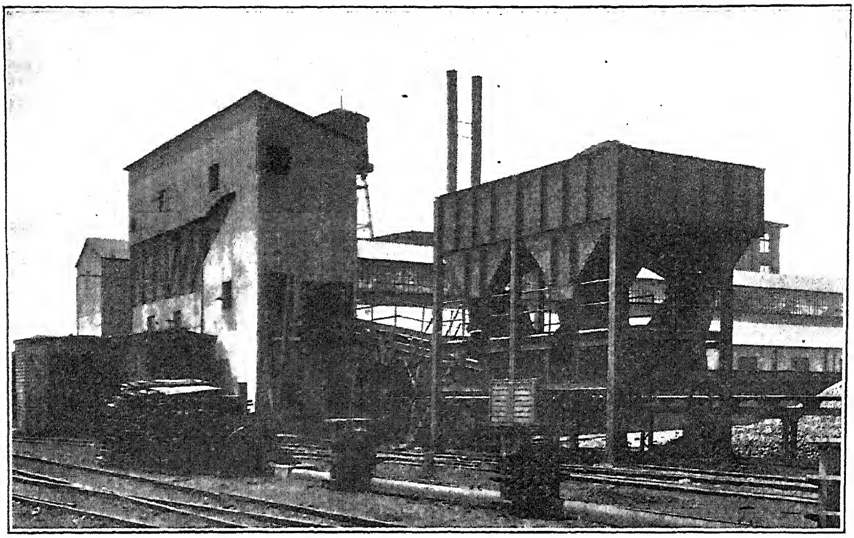


FIG. 2.—BERWIND BRIQUETTING PLANT, SUPERIOR, WIS.

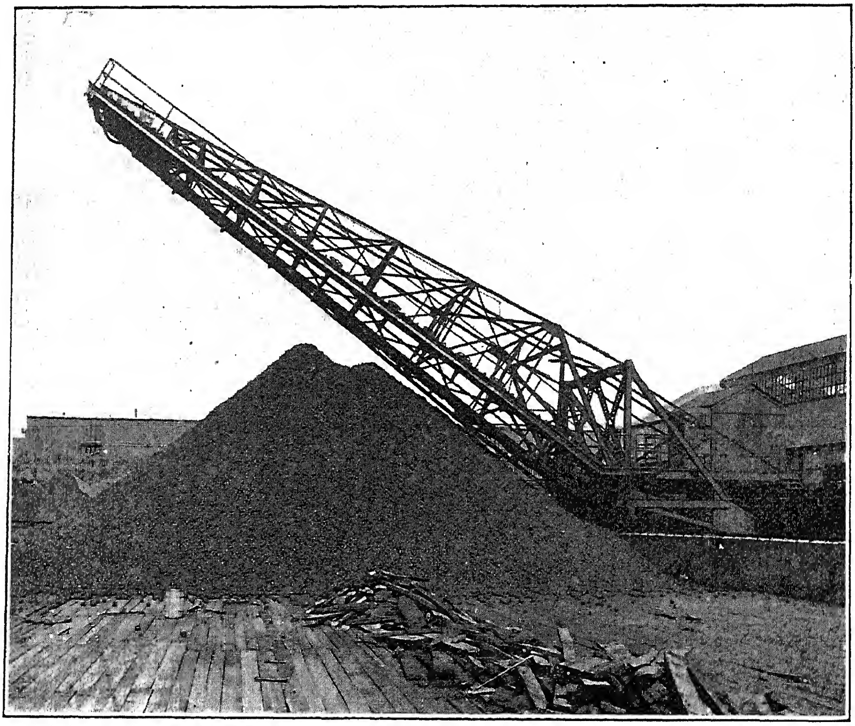


FIG. 3.—BRIQUET-HANDLING EQUIPMENT, BERWIND PLANT, SUPERIOR, WIS.

the plant at Superior may be interesting in this connection. In 1902 approximately 1,000,000 tons of anthracite and less than 10,000 tons of Pocahontas coal had been received at the ports of Duluth and Superior. In 1905 the demand for Pocahontas coal had practically ceased and this condition continued until the Berwind dock was built in 1907. In 1911 these receipts were increased to 1,365,000 tons of anthracite and 520,000 tons of Pocahontas and other "smokeless" coals.

Pocahontas coal is an excellant domestic fuel, but, in order to compete successfully with anthracite, preparation of some kind is necessary. On account of the friability of the "smokeless" coals, the degradation in handling from the tipples in West Virginia to the docks at Superior or Duluth is enormous. It soon became apparent that with over 65

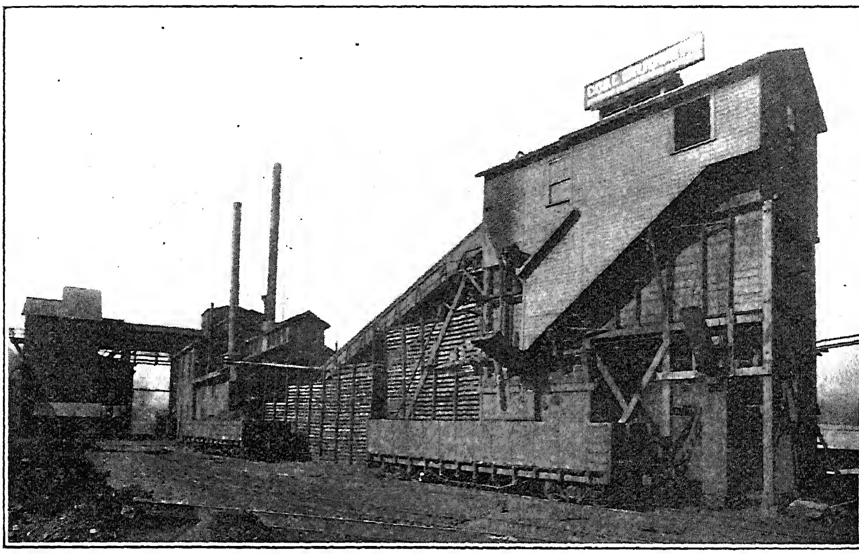


FIG. 4.—STANDARD BRIQUETTE FUEL CO.'S PLANT.

per cent. of the "run of pile" coal passing through a ¾-in. bar screen, either the market for lump and egg sizes would be curtailed or screenings must be moved at a sacrifice in price. Difficulties were also encountered in maintaining the proper demand for the various sizes so that they could be moved as fast as they were prepared and thus avoid the cost and breakage of storing and rehandling. Even with the best preparation and the most careful handling there was always a large percentage of slack in the "egg" and "lump" which the consumer put into his coal bin.

These were the conditions which made the installation of the Berwind plant opportune and profitable. Within a year after the operations were begun a second complete unit was installed, bringing the rated capacity of this plant up to 80 tons of 13-oz. briquets per hour, and making it by

far the largest plant in the country. Thus the briquetting industry at the head of the Lakes was from the outset placed on a sound economic basis (Figs. 2 and 3).

Adequate and attractive advertising brought the merits of briquets before the people of Minnesota and the Dakotas, the principal market for this product. It took some time to convince the dealers that briquets

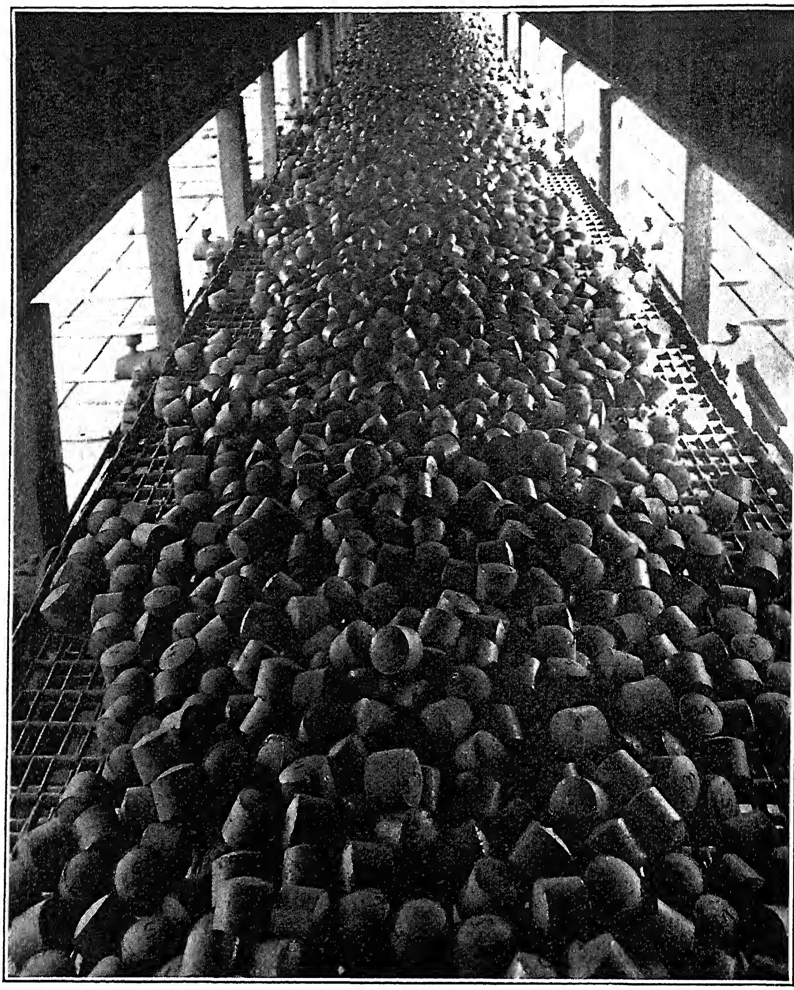


FIG. 5.—BRIQUETS MADE AT STANDARD PLANT.

were cleaner and more economical than the raw coal which they had been buying. The demand for this fuel, however, has steadily increased, which is the best evidence of the continued success of this enterprise. If Pocahontas briquets are not displacing anthracite they are supplying

the demand due to the natural expansion of the country. At the present time briquets are being sold at the same prices as the prepared Pocahontas coal; in this way the market for both products is maintained.

The Standard Briquette Fuel Co. now felt justified in having its plant entirely re-designed and enlarged to accommodate a Rutledge press producing 30 tons of 10-oz. briquets per hour (Figs. 4 and 5). Arkansas semi-anthracite coal is now briquetted with 6 per cent. coke-oven pitch and the briquets are sold in Kansas City and contiguous territory. This is the second season for the new plant, and it is now, for the first time, operating on a successful basis both mechanically and commercially.

Before passing to a description of the plant which forms the basis of this paper, it will not be amiss to mention the Stott Briquette Co.'s plant at Superior, Wis., built in 1910. After several unsuccessful attempts to exploit anthracite briquets, the company decided this year to manufacture a product consisting principally of Pocahontas coal. A modified Belgian roll press built by the Mashek Engineering Co. is used, producing 2-oz. pillow-shaped briquets at the rate of 12 tons per hour. The residue from petroleum is used as a binder.

In all coal briquetting the binder is a factor of prime importance. Coal-tar pitch has been generally recognized in Europe as the logical binder for coal briquets. It was adopted in this country with the industry. There are many admirable reasons for its use. It is cheap, easy to transport and to handle in the manufacturing process, is waterproof, acts as a preservative to the coal, and produces coke from an otherwise non-coking coal. It is a coal product, and if properly incorporated with the coal becomes a part of it during combustion. When hard pitch is used in dry form, the briquets thus made with bituminous coal produce less smoke than the raw coal. It has one serious defect: it will not make a smokeless fuel out of anthracite culm.

To remedy this evil, binders of vegetable origin have been much exploited. They are, however, expensive, difficult to handle, and do not make waterproof briquets. Two small plants in the country are using vegetable binders of which starch is the principal ingredient, and one plant uses oil emulsified with starch. In all cases the briquets must be baked before shipment and the product is irregular in quality. The processes can hardly be said to be commercially successful.

Another excellent binder is asphalt obtained from the distillation of petroleum. It has many of the qualities of coal-tar pitch; the difficulties in its use will be referred to later.

There is a steady and growing demand in the Pacific Coast States and in British Columbia for coals mined in that region, although the higher-grade coals from Wyoming and other Rocky Mountain States also reach this market. Until a few years ago the demand for steam and

domestic coals was about equal, so that the coal operators could market their prepared coals without throwing an undue surplus of any one size on the market. The enormous increase in the production of petroleum in California, however, necessitated the creation of an adequate market, and the price of fuel oil was therefore reduced in order to move this production. For industrial plants and railroad purposes steam coal was rapidly displaced by the economically superior oil, so that by the end of 1913 practically all the furnaces in the locomotives, steamships, and large steam plants were transformed to burn oil.

The Pacific Coast Coal Co., as the largest producer of coal on the Pacific Coast, early realized what this condition would eventually mean, and began to search for some means to convert its now unmarketable fine coal into a profitable domestic fuel. The late James Andersen, Chief Engineer of the company, was delegated to make this investigation. After he had visited many briquetting plants in the United States and Europe covering a period of three years, the Malcolmson Briquet Engineering Co., of Chicago, was awarded a contract to design and build a complete coal-briquetting plant.

At Black Diamond, Franklin, and Burnett, a good grade of bituminous coal, relatively low in ash, is mined by the Pacific Coast Coal Co.; the New Castle coal, however, falls within the government classification of sub-bituminous coals. While the larger sizes furnish an excellent domestic fuel, for which there is a ready sale, the New Castle fine coal was the first to be displaced by the introduction of oil. This coal will not coke, but this objectionable feature is overcome by the admixture of a small percentage of the company's South Prairie screenings, which produce a strong coke. The result is a highly desirable domestic fuel, meeting all the requirements of a merchantable briquet.

The coal-briquetting plant of the Pacific Coast Coal Co. is located near Seattle, Wash., close to the shore at the southern end of Lake Washington. It is an ideal location for a manufacturing plant. A virgin forest was cleared in preparing the site. An abundance of water is pumped from the lake at small expense. An equable and salubrious climate lends itself to the efficient and continuous operation of the plant (Fig. 6). The plant is served by the New Castle branch of the Columbia & Puget Sound Railroad, from which a spur passes the plant on its way to the coal piers at this end of the lake.

The raw coal is delivered to the plant in dump-bottom cars, and is unloaded through a track hopper alongside the raw-coal bin. A gravity-discharge elevator, into which the coal is delivered from the track hopper by a reciprocating feeder, elevates the coal to an 800-ton raw-coal storage bin, of wooden construction and divided into two compartments.

Two feeders deliver the coal from these compartments into two flight conveyors. These feeders are designed for close regulation so that the requirements of the plant can be adjusted at this point. The feeders are driven through friction clutches by an electric motor.

The raw-coal conveyors travel horizontally and terminate at the intake chutes of the driers. Between the driers and the raw-coal bin a Williams dustless crusher is installed. Gates and chutes permit the delivery of coal from either raw-coal conveyor to the crusher, from which it is discharged into a continuous-bucket elevator and is again elevated to the raw-coal conveyors.

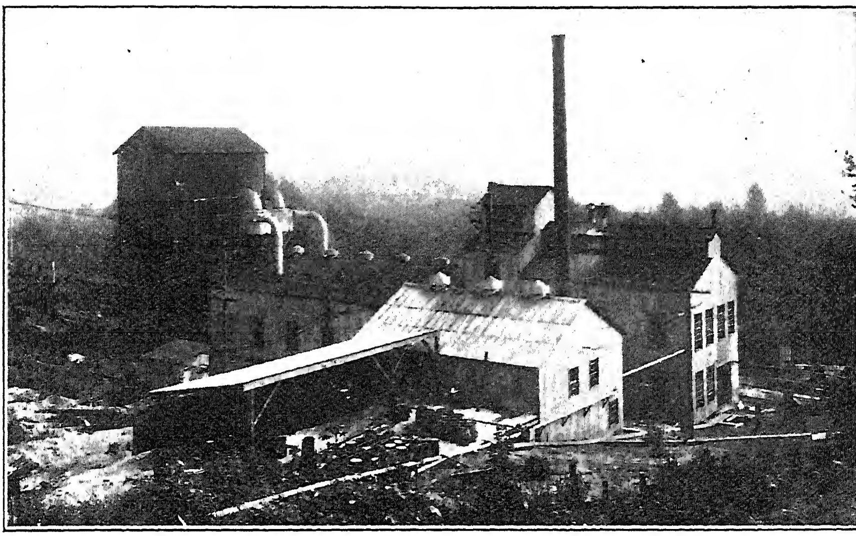


FIG. 6.—BRIQUETTING PLANT OF PACIFIC COAST COAL CO., NEAR SEATTLE, WASH.

If the coal is very wet, it may be passed directly through either drier to a flight conveyor installed between the driers, which returns it to the crusher. The necessary part of the surface moisture is eliminated in the first drier, and after crushing the coal is completely dried and heated in the second drier.

This duplicate arrangement of driers and conveying machinery permits the blending of two or more coals to improve the quality of the mixture, and insures uniformity of the dried product.

Two Ruggles-Coles A–14 driers of improved design, with shell of special length, are installed in the drier building (Fig. 7). Each of these driers is direct-driven by an electric motor of sufficient size to pick up the load under all conditions. The exhaust fans are driven by variable-speed motors through silent-chain drives, and the exhaust gases pass through

cyclone separators located above the roof of the drier building. The dust from these separators is returned to the raw-coal conveyors.

Fuel oil furnishes the heat for drying the coal, and is introduced into the driers through Billow furnaces built by the National Supply Co. of

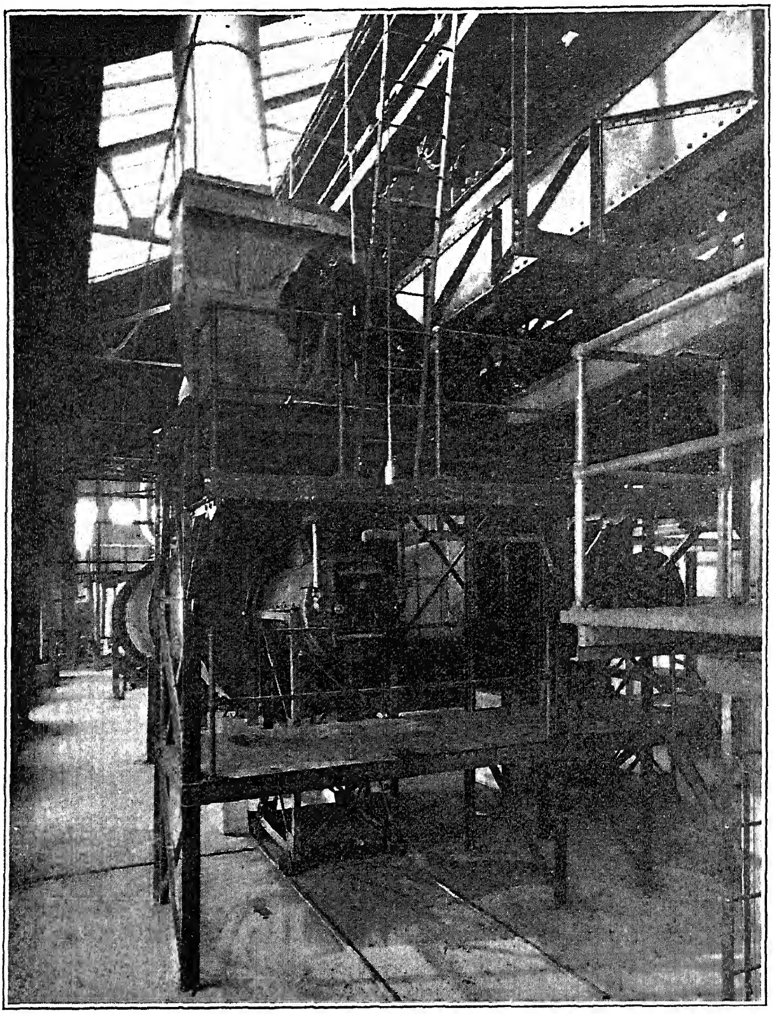


FIG. 7.—DRIER BUILDING, PACIFIC COAST COAL CO.

Chicago. A General Electric centrifugal air compressor furnishes air for atomizing the oil, and perfect combustion is thereby obtained. These are the most satisfactory furnaces yet installed by us for drying coal, in that they permit very close regulation and a uniform temperature of the dried coal at the discharge end. The furnaces are carried on structural-steel

frames supported on wheels, so they can be removed without much expense whenever it is necessary to enter the driers for repairs.

The dry coal is taken from the driers by means of a continuous-bucket elevator, to a small dry-coal bin, located directly under the roof of the machine building. All of the equipment, from the raw-coal feeders to and including the dry-coal bin, is of steel construction and absolutely dust tight. This feature increases the efficiency of the drying system by preventing the ingress of cool air, as well as insuring a dust-free atmosphere in the buildings.

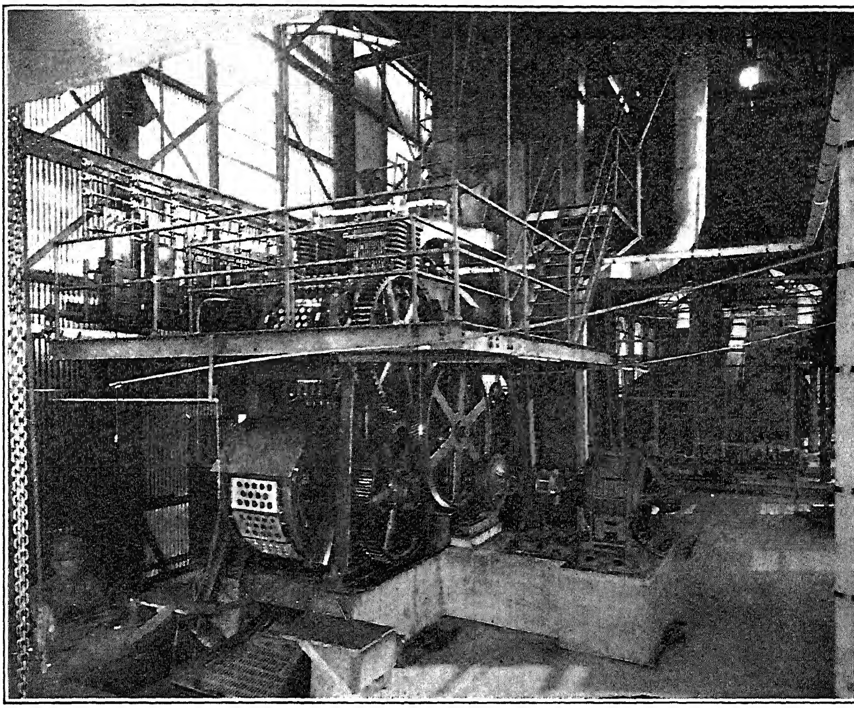


FIG. 8.—RUTLEDGE BRIQUETTING PRESS AND FLUXER.

Underneath the dry-coal bin an apron feeder is located, which serves to regulate the quantity of coal required by the briquetting press, and also to deliver this coal to the preliminary mixing equipment.

Liquid binder is maintained at constant pressure and uniform temperature in a small storage tank, located directly above the mixing equipment, and is introduced through a Forman steam-jacketed regulating valve into the mixer containing the dry coal. A uniform flow of binder is insured by this equipment and the proportioning of binder and coal is under the immediate control of the operator on the charging platform. Taylor index thermometers located here indicate the temperatures of

coal and binder; and the dust-collecting system installed above this equipment removes all dust caused by the handling of the dry coal.

After the coal and binder have been subjected to a preliminary mixing they pass in a uniform stream to the Rutledge fluxer (Fig. 8). This piece of special machinery is the result of long experimentation in the development of equipment for handling and mixing large quantities of coal and binder at relatively high temperatures. A thorough blending or "fluxing" of the coal and binder is of prime importance in obtaining satisfactory and uniform briquets at low cost. A simple mechanical mixing of the raw materials will not achieve the desired end; a thorough coating of the coal particles with a thin film of binder, so intimately associated as to approach a chemical union, is the result striven for and obtained by this efficient equipment.

The Rutledge fluxer consists essentially of a large chamber and two partly inclosed superimposed sections, where superheated steam is introduced into the mixture through a carefully worked out piping system. These horizontal shafts are provided with a series of paddles so arranged as to agitate and thoroughly flux the coal and binder as it passes downward through the successive chambers. The temperature of the agglomerated mass is raised at a uniform rate and with greater economy in the successive stages owing to the partly inclosed chambers. The fluxer is operated through a train of gears driven by an electric motor through a silent-chain drive and friction clutch.

The use of asphalt obtained from the distillation of petroleum as a briquet binder has been perfected through a process invented by Robert Schorr. This process was developed in the operation of several small plants which were built some years ago in California under his direction. The briquetting of coal with this binder has never before been attempted on such a large scale. Considerable time was spent in adapting the Schorr process to the conditions obtaining at the Seattle plant. The results finally produced are an indication of the excellence of this process and prove it indispensable in the manufacture of briquets using asphalt as a binder.

The Rutledge briquetting press installed in this plant is of the same type as those used in the plant of the Berwind Fuel Co. at Superior, Wis., except that it manufactures a smaller briquet and incorporates the improvements which grew out of the operations at Superior (Fig. 9).

The Rutledge press has been designed to meet American conditions by producing a large output of relatively small briquets under high pressure with a small percentage of binder. The handling of 30 to 40 tons of coal per hour through a manufacturing plant involves problems which are not anticipated by the experience in briquetting 8 or 10 tons per hour, the usual output of the other machines on the market.

The Rutledge press is of the continuous-mold type. It consists

essentially of a chain of molds or die plates, passing under a feeding hopper and between two revolving drums, upon which punch rams are carried. Each die plate contains 14 dies. Twelve punch rams are mounted on the upper and 12 on the lower drum; each punch ram supports 14 punches. Two pilot punches on each punch ram engage the die plates in advance of the entrance of the punches into the dies; these pilot punches not only serve to move the dies forward, but insure perfect

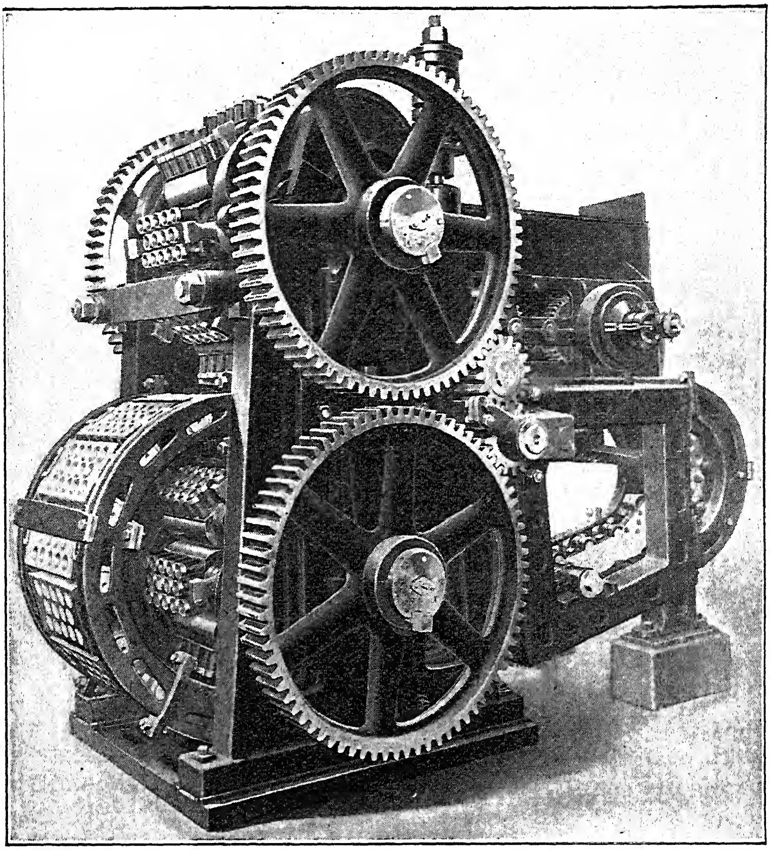


FIG. 9.—RUTLEDGE BRIQUETTING PRESS.

alignment of the dies and punches. The lower punches enter the dies again at the lowest point of their travel, and eject the briquets.

The punch rams oscillate on seats in the drums; the movement is produced by arms, cast on the rams, traveling in fixed cam tracks. This movement is so timed with relation to the angular displacement of the punch rams about the main shafts upon which they are mounted, that the punches enter the dies and compress the briquets with a straight-line motion.

The bearings of the upper drum are set in vertical guides, and operate against heavy helical springs. These springs are set to move upward when the pressure of the punches on the briquets exceeds 4,000 lb. per square inch. Under normal working conditions the springs compress slightly at every compression of the punches. The briquets are cylindrical in shape, with spherical ends, and weigh 10½ oz. A diamond stamped on one end identifies the "Black Diamond" briquet. For the briquetting of New Castle coal, punch tips containing the letter "N" have been provided. The press is turning out 32 to 33 tons of briquets per hour.

The Rutledge press and fluxer are built entirely of steel, with the exception of the bearings, which are of special bronze alloy. The parts receiving the most wear, such as the die bushings, cam track, and punch-ram guide rollers, are of chrome-nickel steel, heat treated, or manganese steel, and are easily replaceable.

The briquets are discharged from the press on to a perforated chute which delivers them to the cooling conveyor (Fig. 10). All of the "fines" from the press and any damaged or inferior briquets pass through this chute into a "refuse" flight conveyor. This conveyor passes underneath the press and delivers its charge into the elevator which returns it to the fluxer.

The cooling conveyor is composed entirely of link-belt malleable chain. The head shaft carrying the sprockets is driven through the necessary gears by an electric motor. This conveyor passes horizontally below the floor line, from the front of the press to the outside of the machine building; whence it travels a short distance on a slight incline following the rise of the ground; and again horizontally on a wooden trestle until it terminates at the lower floor of the briquet loading pocket. The speed of the conveyor is fixed so that it may be loaded uniformly and the briquets may have the greatest opportunity for cooling. The surface cooling of the briquets is also assisted by the use of water in a fine spray delivered from nozzles installed along the course of and above the conveyor.

The briquet loading pocket is a wooden structure of small capacity. It is not intended for storage purposes, except to receive the briquets during the shifting of cars. While cars are being loaded the briquets are drawn from the pocket through duplex undercut gates into a gravity-discharge bucket elevator, which also serves to convey the briquets from the end of the cooling conveyor to the loading chute, or elevate them to the briquet pocket, as desired.

The asphalt used as binder comes from California, and is received in barrels on a spur of the main line of the New Castle branch at an elevation of about 80 ft. above the foundation line of the briquet plant. The barrels are skidded on a chute to a small storage yard adjacent to the

boiler and melting-kettle house. Here they are stripped as required, and the asphalt is charged by hand into the rear end of the kettles.

There are two melting kettles, of the same size and design, built with shells of heavy sheet steel hung from structural-steel frames. Fuel oil furnishes the heat for melting the asphalt. Care has been taken in the design of the furnaces to distribute the heat uniformly, and prevent the usual difficulties due to intense local heating. These furnaces are also

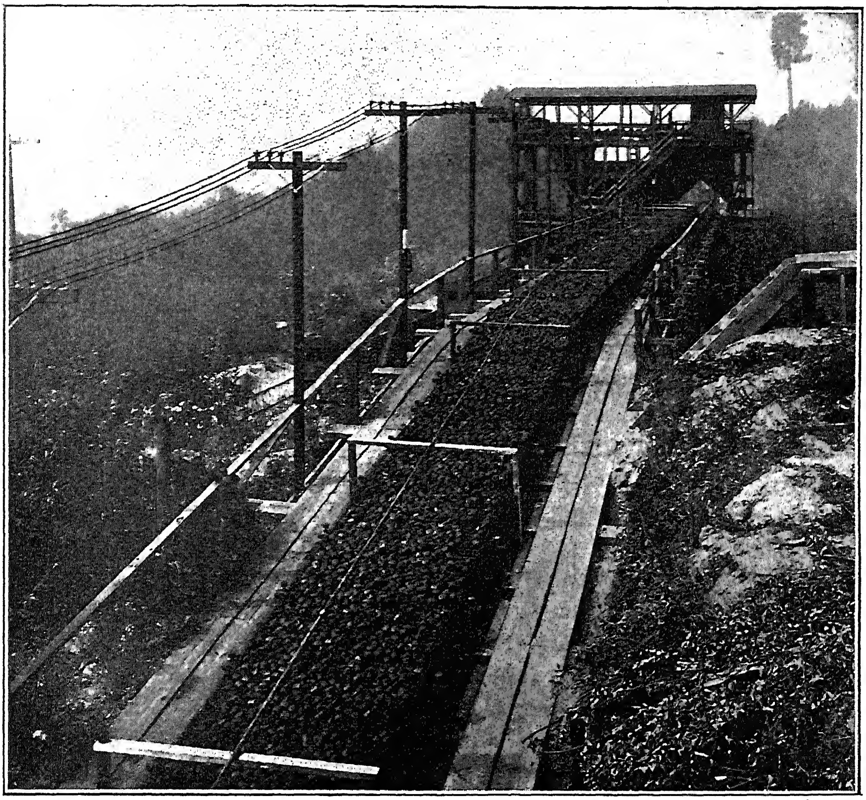


FIG. 10.—COOLING CONVEYOR.

designed for the greatest fuel economy, in that the gases from one furnace may be passed through the other furnace before reaching the stack. Provision is also made to heat the incoming air with the exhaust gases. Billow oil furnaces, similar to the drier furnaces, are used. A stirring device directly driven by a vertical motor is placed in one kettle. The usual procedure is to melt the asphalt in one kettle and retain the other full of melted asphalt at the proper temperature, as a reservoir, from which it passes to the pump.

The intention of the Pacific Coast Coal Co. is to erect eventually a

complete distilling plant and manufacture its own binder. It is for the purpose of consuming the distillates from this process that oil-burning instead of coal furnaces have been installed.

In front of the melting kettles a Kinney rotary steam-jacketed pump is installed, together with an electric motor which operates the pump through a silent-chain drive. The piping on the front of the kettles is so arranged that the asphalt can be drawn from the bottom or side of either kettle to the pump. The hot asphalt is pumped directly to the small reservoir above the charging platform of the briquet-machine building, and any overflow from this reservoir is returned to the melting kettles. All of the asphalt pipes are incased in asbestos covering and are steam-jacketed to insure continuous operation.

The fuel oil is delivered to the briquet plant in tank cars and discharged into a reservoir located beside the upper unloading track, from which it passes to the Billow oil-pumping system in the boiler and melting-kettle house. This equipment strains and heats the oil and pumps it to the various points of consumption under uniform pressure. The system is automatically controlled; and to its efficiency may be largely attributed the successful operation of the oil-burning equipment. All of the oil pipes are insulated and heat-protected.

One 72 by 16 horizontal return-tubular Wickes boiler furnishes steam for the fluxer, the operation of the steam pumps, and if necessary the oil burners. Oil is also used for boiler fuel.

The main buildings are built entirely of structural steel with corrugated sheet-steel siding and asbestos-protected metal roofs. Steel-sash ventilators with mechanically operating devices are used in the machine building, and star ventilators in the drier building and boiler house. All the machinery supports and stairways are of structural steel, and the main platforms around the press and drier are of checkered plate. The only wood used in the main buildings is on some of the upper platforms and the windows and doors.

All the conveying and elevating machinery was manufactured by the Link-Belt Co.; it is of ample capacity and of very heavy construction, including steel gears, sprockets, and driving chain. As far as possible, machinery of standard design has been utilized or modified to meet the special requirements of the plant. This reduces the cost of repair parts and makes them readily obtainable.

An independently fired Foster superheater built by the Power Specialty Co. is installed in the corner of the machine building opposite the briquetting press. A small Billow oil-burning furnace furnishes the necessary heat.

General Electric electrical equipment is used throughout. Slow-speed motors and individual drives are installed, and in most cases direct

connected. Wherever the necessary reduction in speed is great, silent-chain drives are used.

The control of all electrical equipment is concentrated at two points in the drier and machine buildings. Pipe boards, built by the General Devices & Equipment Co., carry all of the control apparatus for these motors. Two panel boards distribute the current for the electric lighting system. All electric wires are carried in steel conduit, and the latest approved type of safety devices is used.

Three-phase, 60-cycle alternating current generated by the hydro-electric station at Snoqualmie Falls is purchased of the Puget Sound Traction, Light & Power Co., and is delivered at 2,300 volts to the transformer station at the briquet plant. It is here reduced to 440 volts for power and 110 volts for lighting service. A pole line was built by the Pacific Coast Coal Co. to transmit this current from the substation at Renton.

A complete water system has been put in by the company to protect the auxiliary buildings of the plant which are of wooden construction. A small centrifugal pump, driven by an electric motor with remote control, pumps the water from Lake Washington to a small reservoir located directly in front of the main building. Here a high-pressure steam-driven Underwriters' pump is installed to pump the water to a 50,000-gal. tank on the hill beside the oil reservoir. Hydrants are located at all desirable points.

A description of this necessary and valuable adjunct to the coal properties of the Pacific Coast Coal Co. would not be complete without reference to the assistance rendered by N. D. Moore, Chief Engineer of the Pacific Coast company, and Ralph Galt, now Superintendent of the plant, under whose direction the building and installation work was carried on.

The plant has been in operation since the latter part of August, turning out merchantable briquets, which have been distributed to the local trade and neighboring cities in order to satisfy as far as possible the demands for this fuel. Advices from the sales department indicate that the briquets are everywhere being received with satisfaction. They are sold in competition with the best grades of bituminous coal produced in the Pacific Coast States and Canada. The fact that they are sold in Victoria and other British Columbia cities in competition with the high-grade bituminous coals produced on Vancouver Island is the best evidence of the superiority of briquets over coal in its raw state.

The Pacific Coast Coal Co. has made the necessary preparations to double the capacity of this plant. The second unit and the necessary auxiliary equipment will be installed as soon as it becomes evident that the permanent demand for briquets has reached the maximum output of the present plant.

Discussion

E. W. Parker, Washington, D. C.—The briquet or boulet, or coalette (whatever name is used) has come to stay as a part of the fuel supply, particularly for domestic purposes. During the past winter I have had some personal experience with the use of boulets made from anthracite coal. All of this winter I have used boulets in my furnace and in my grate at my home in Washington. One reason for doing so is that we get the boulets at a price considerably cheaper than the prepared sizes of anthracite coal. The fuel has proved entirely satisfactory. One advantage possessed by the boulet is its absolutely uniform size, which makes the ventilation of the firebed practically perfect. The boulets burn until entirely consumed. There is not, even in an open grate fire, if the fire is kept constantly going, any unconsumed fuel passing through the grate bars. The boulets hold their form until entirely consumed and then break away into a clean white ash, with absolutely no clinker. That in itself is a particularly favorable recommendation to the housekeeper.

Mr. Malcolmson has mentioned the fact that notwithstanding the smokeless character of anthracite, the boulets made from it emit smoke. The emission of smoke is due to the coal-tar pitch used as a binder. This is particularly noticeable if the fireman is somewhat careless, or wants to save labor, and feeds too much of the fuel on the fire at one time, cooling it down and thus causing an unnecessary amount of smoke. The smoke lasts but a short time, however, and if the fire is fed a little more frequently, with a small quantity of fuel at one time, the smoke is reduced to a minimum. I believe it will not be long before some of our engineers will have overcome the objectionable smoke, and will produce either from anthracite or from the so-called smokeless New River and Pocahontas semi-bituminous coals, a briquet which will be as smokeless as anthracite for all practical purposes.

Mr. Malcolmson in speaking of some of the financial wrecks that have marked the progress of the development of the briquetting industry has omitted to mention one factor which, I think, has had more influence in this respect than any other. This has been the endeavor to exploit secret or patented processes and binders which, when shown to the over-anxious prospective investor, showed very remarkable results as a fuel. The briquets burned well and appeared to carry out the claims made by the promoters, but when put upon a practical basis for commercial operation the projects have resulted in failure.

There is, on the Pacific Coast, an interesting development in the manufacture of briquets not made from coal at all. The ordinary manufactured fuel and illuminating gas in Los Angeles, Cal., and Portland, Ore., is made from petroleum. The residue consists of material which is called

carbon, as it really is. At Los Angeles for a number of years the Los Angeles Gas & Electrical Corporation has been briquetting this residue, and making a very acceptable domestic fuel in briquetted form. A former waste product has been converted into an excellent fuel. Within the last year a similar plant was erected at Portland.

ALFRED C. LANE, Tufts College, Mass.—Is there any probability or any possibility that the method of using coal dust directly, in producing gas, or as a fuel, might use up all the coal dust, without briquetting?

E. W. PARKER.—I hardly think so. Some powdered fuel is used in cement manufacture, in which case the ash in the coal goes into the clinker and really adds to the output of cement. For boiler practice, however, the use of powdered fuel has not, so far as I know, proved successful. The intense heat created necessarily fuses the ash, and it is deposited as obsidian on the bridge walls, boiler tubes, etc. It makes as perfect an insulation as one could possibly desire.

A. C. LANE.—I had reference also to the system developed by Pintsch and Diesel, where the anthracite coal is burned and the fuel gases are recovered and used in a reservoir.

E. W. PARKER.—I do not know of any really successful use of powdered fuel in the manufacture of producer gas or other gases. Some of the smaller sizes of coal are used for that purpose, but at Point Breeze, at the lower part of Philadelphia, the United Gas Improvement Co. for a number of years has briquetted fine coal, both anthracite and bituminous, with a mixture of coke breeze. Water gas was made from the briquetted fuel.

CHARLES DORRANCE, JR., Lansford, Pa.—In regard to the use of powdered anthracite as a fuel, I would say that for Mr. Lathrop, of the Lehigh Coal & Navigation Co., a number of years ago, I ran a test on a Barnhurst furnace burning pulverized anthracite coal. The coal, anthracite culm or slush, was pulverized in a cement mill, through 80 mesh, and was then blown through a tuyère into a specially constructed combustion chamber, so designed as to give a radial motion to the flame and fuel as it went in. The combustion chamber was heated first with bituminous coal, and after the temperature of the combustion chamber was high enough, the anthracite coal was introduced instead of the bituminous, and a very intense blast-furnace-like heat was maintained. We ran two 24-hr. boiler tests on that and developed some hours as high as 50 per cent. over the rating of the boiler. The main trouble in the whole process seemed to be that the refractory which formed the lining of the combustion chamber would not stand the blowpipe action of the flame, and the scouring action there would eat the lining out in a few weeks' time.

The other chief disadvantage of the process was pulverizing the anthracite coal. Pulverizing bituminous coal is simple, and is done all over the cement region to-day, but the anthracite coal, having a cubical fracture, does not pulverize to dust, but goes down on to its different cleavage planes smaller and smaller, and we figured out that to pulverize anthracite successfully on a commercial basis would cost in the neighborhood of 20c. to 30c. per ton. When you figure that the price of No. 3 buck or barley anthracite coal now is anywhere from 30c. to 50c. a ton, the feasibility of pulverizing anthracite coal as a fuel is doubtful. I have also the statement of a mechanical engineer of the biggest cement plant in the Lehigh region, in fact in America, that they have spent a great deal of money in trying to use powdered anthracite coal in their cement industry, and have given it up as being very much more expensive than either run-of-mine bituminous or pulverized bituminous coal would be.

Edwin Ludlow, Lansford, Pa.—In regard to the briquetting of anthracite coal, the Lehigh Coal & Navigation Co. built a briquetting plant several years ago, and has been manufacturing briquets ever since, and produced the boulets or briquets that Mr. Parker has just spoken of.

The briquets are made of 93 per cent. of anthracite culm and 7 per cent. of dry pitch binder; the materials are heated by steam, and thoroughly intermixed in a pug mill until the mixture has what we call a "blue mix color;" it is then run through a press, making boulets of a size intermediate between nut and stove coal, especially adapted for domestic purposes.

Although we sell these boulets at $1.50 to $2 a ton less than nut coal, we have found no very large demand for them; in fact, it is difficult to get repeat orders on account of the smoke in the binder. This smoke is not troublesome to the man using the boulets, as by keeping the draft on for not more than 5 min., when the fire first starts, the smoke is burned off and the boulets then make a bright, clear fire superior to anthracite for an open grate or for a stove. But this smoke is very heavy, with a pungent odor, and after passing out of the chimney of the man who is burning the boulets it descends into the windows of the man next door, and the objection to the use of boulets comes from the neighbor, rather than the man who is burning them.

In order to overcome this objection, we have been manufacturing boulets with a smokeless binder that gives equally good burning results and absolutely no smoke; the objection to these boulets is that they are not waterproof, and will soften if exposed to the weather. Mr. Malcolmson is carrying on experiments with the weatherproofing of this class of boulets and assures us that it is entirely practicable, and we

propose trying out these experiments on a commercial scale to test the laboratory results that Mr. Malcolmson has obtained.

It is perfectly true, as Mr. Malcolmson states, that the anthracite companies as a rule are not giving the question of briquetting anthracite culm very serious attention at this time; the last two years having been years of slack demand, so that the anthracite companies have not been able to run full time and all of them have accumulated large stocks of prepared sizes. It would not be policy for them to decrease the sales of their own prepared sizes by manufacturing a fuel from a waste product to compete with them.

The anthracite companies, however, are prepared for the day that we all look forward to, when the demand for the prepared sizes of anthracite will exceed the production of the collieries, and when that day comes, the companies will be found with large stocks of culm ready to be made into briquets as fast as the trade will take them.

Formerly, anthracite culm was thrown on the waste banks with the breaker slate, but at the present time most of the companies are separating their culm, and where it is not being used for slushing operations underground, it is being stored in preparation for the day when there will be a market for it; but until the market will absorb the prepared sizes it is obviously better for the companies to store their waste product of slush rather than their prepared sizes, that represent an investment of from $3 to $4 a ton.

W. H. Blauvelt, Syracuse, N. Y. (communication to the Secretary*).—Mr. Malcolmson's paper is of special value to those who are interested in the briquetting industry either on account of its important relation to the conservation of the fuel supply of our country or for more personal reasons; because it describes several plants which have been established on a sound commercial basis and give every indication of permanent and successful operation. Unfortunately, the record of the undertakings for the briquetting of coal in this country has been heretofore practically one of unsuccessful attempts. Failures have been due in most cases to lack of appreciation of all that goes to make a successful briquetting operation. In several cases only one or two seemingly minor operating details prevented success. Other installations failed on account of lack of suitable market. Either there was not enough difference between the price of the raw material and the finished product, or there was no place for the new fuel in competition with other fuels already in use. It is satisfactory to note that in the plants described by the author of the paper all the conditions seem to have been met and the plants apparently have come to stay. With these installations to open the way, it is reasonable to expect that other plants will follow very soon.

* Received Feb. 15, 1915.

I am especially interested in these satisfactory installations on account of my own experience with a briquetting plant, which I reported to the Institute in a paper read at the Pittsburgh meeting in 1910. This paper described a plant built by the Solvay companies at Detroit primarily for the manufacture of briquets from coke breeze from their coke ovens. The plant described in my paper was worked out to a commercial success, so far as the cost of manufacture and quality of product were concerned, and a very satisfactory product was marketed for some time at a profit. After my paper was presented some experiments were carried out in the baking of the briquets in order to make them smokeless. These experiments were entirely successful. Sufficient coal was mixed with the coke breeze so that with the pitch binder there was enough cementing material to make a strong briquet, which produced no smoke in burning, retained its original form, and yet was coherent enough not to break down in handling. But after the briquets began to be thoroughly established in the open market it was found that they were competing directly with another product of the company, namely, domestic coke, so that every ton of briquets that was placed meant the sale of one less ton of domestic coke. As the result of this entirely unforeseen condition the briquetting plant has not been in operation for several years. This experience is an excellent illustration of the careful study that must be given to all the conditions before undertaking the manufacture of briquets.

The author's description of the successful use as a binder for briquets of the asphalt obtained from the distillation of petroleum oils of asphalt base is of special interest, particularly in connection with briquetting plants on the Pacific Coast. Coal-tar pitch is the only other binder that has been generally successful either in this country or Europe, as practically all the other binders either fail to make a waterproof briquet, or else they add materially to the percentage of incombustible matter in the finished product. It would appear that America cannot much longer postpone the utilization of coal tar in the manufacture of the multitude of useful products of which it is the base. When these industries have achieved the important position in America which they deserve, and which they could occupy under the protection of the intelligent legislation which we hope will grow out of the lessons we have been learning during the past six months, let us hope that the briquetting industry will also have developed to a point where it will offer a sufficient market for the pitch which is the residue from the first step in the manufacture of the coal-tar products.

Underground Haulage by Storage-Battery Locomotives in the Bunker Hill & Sullivan Mine

BY J. W. GWINN, KELLOGG, IDAHO

(New York Meeting, February, 1915)

THE underground haulage system in the lead-silver mine of the Bunker Hill & Sullivan Co., situated at Kellogg, Idaho, is the most extensive in the Cœur d'Alène district, comprising about 35,000 ft. of tunnels, drifts, and cross-cuts where locomotive haulage is in use. The main working adit, known as the Kellogg tunnel, or No. 9 level, is approximately 10,000 ft. in length from the portal to the main working shaft and extends beyond that about 4,500 ft. in each of two directions. On this level all the ore from the Bunker Hill & Sullivan, the Sierra Nevada Consolidated, and the Caledonia Mining Co.—an average of 44,600 tons per month—is hauled to the mills, located a short distance from the portal. Electric haulage by the trolley-type locomotive is used on this level throughout.

Below this level, at intervals of 200 ft. vertically, are five other working levels (Nos. 10, 11, 12, 13, and 14); and the total haulage distance on these, where electric locomotives are used, is approximately 16,000 ft. No. 13 level has but recently been opened up and is not working to capacity; while the main cross-cut on No. 14 has just been started, and has no need of power haulage. Of the four levels equipped with electric haulage, Nos. 11, 12, and 13 have storage-battery locomotives, and No. 10 still retains the trolley type by reason of the small and decreasing tonnage produced.

Electrification

Electric power is supplied to the mine by two systems. The first, the 500-volt direct-current trolley, was installed in 1897 when the Kellogg tunnel was started, and was extended as the tunnel advanced, being used for the first trainload of ore on Nov. 17, 1902, since which time it has been in continuous service. The current for the line is transmitted on a No. 00 copper wire as far as the shaft, and on No. 0 wire throughout the remaining length of the Kellogg tunnel; and until the installation of the storage-battery locomotives it was used in practically all the cross-cuts and drifts on each level below No. 9.

The second circuit, 2,300 volts, alternating current, was installed in 1907, to furnish power for the pumps, as well as lights for the shaft and stations, and for preheating compressed air for the hoist. This current is transmitted on a No. 2 B. & S. gauge, triple-conductor, varnished cable incased in a lead sheath. The installation of an electric hoist in November, 1911, increased the load on the cable so much that an additional cable, No. 000, leaded, varnished cambric, covered with jute, was added, as an additional protection is case of cable break-downs. The two cables are kept continuously in service, so that any of the motors requiring 2,300-volt three-phase service can be supplied from either of the two. Also, any break-down in insulation, or other injury, can be immediately detected and repaired before any possible call can be made on either of

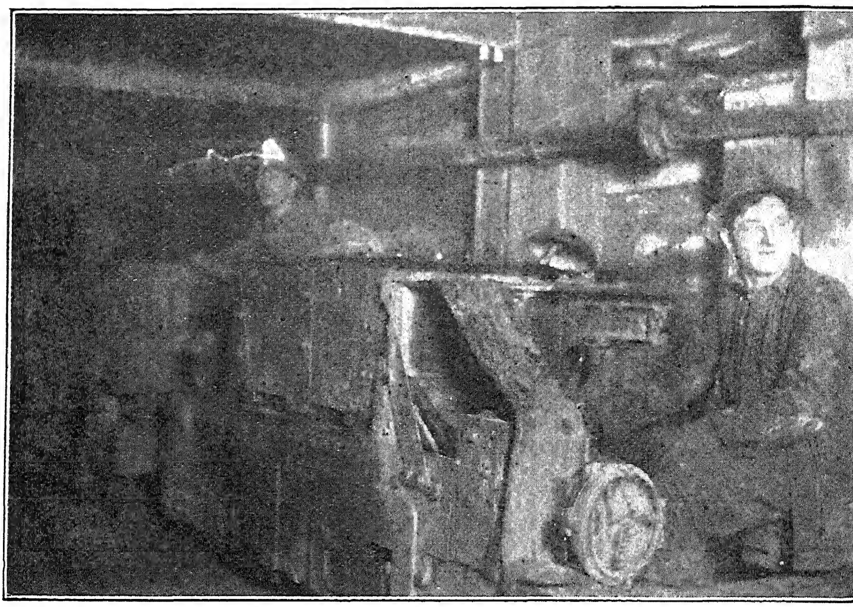


FIG. 1.—2½-TON JEFFREY LOCOMOTIVE ON NO. 11 LEVEL, SHOWING THE ARRANGEMENT OF THE INTERCHANGEABLE AUXILIARY BATTERY OF CELLS.

them for the full load of the pumps and hoists combined. This practically insures continuous service so far as the transmission of current underground is concerned. In addition to being used for the purposes stated above, these high-voltage lines also furnish current for the motor-generator set used in charging the storage batteries.

Storage-Battery Installation

The first storage-battery locomotive to be installed, replacing a 2½-ton Jeffrey trolley-type, was a Jeffrey, put into operation Mar. 3, 1913, on No. 11 level, of the following specifications:

Weight	5,000 lb.
Maximum volts required at battery terminal for charging	125
Average volts at battery terminal on discharge	75
Number of motors	2
Wheels inside or outside	inside
Approximate full-load speed, per hour	4 miles
Track gauge	24 in.
Overall width	40 in.
Length of frame	83 in.
Height of frame above rail	24½ in.
Height over battery	46 in.
Wheel base	27 in.
Diameter of wheels	16 in.

The battery equipment consists of 63 cells of Edison type "A 4," having a capacity of approximately 112 ton-miles on a single charge. Ampere-hour capacity, 150.

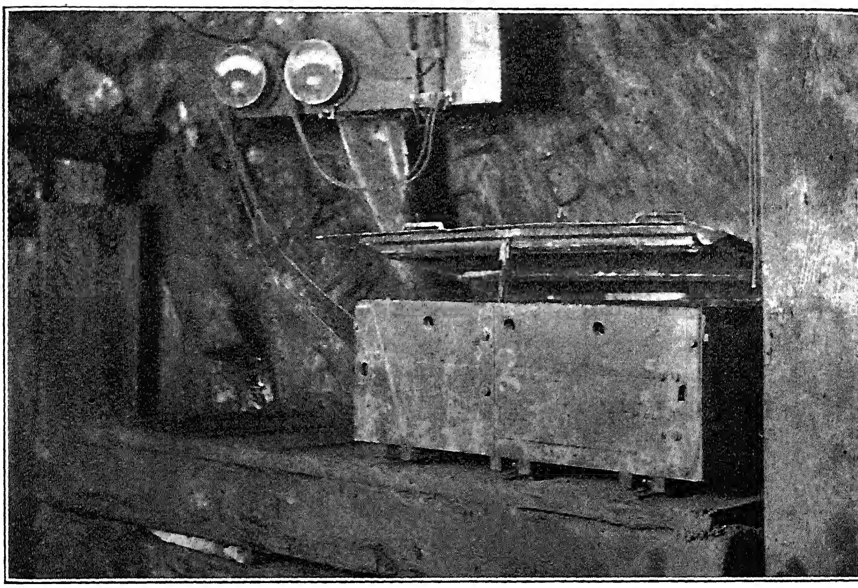


FIG. 2.—AUXILIARY BATTERY OF CELLS FOR 2½-TON JEFFREY LOCOMOTIVE BEING CHARGED AT THE STATION ON NO. 11 LEVEL.

The locomotive was equipped with two batteries, so that one could be left at the charging station to be charged, while the other was on duty. This was made necessary by the insufficient capacity of one battery to run for the required time without loss of time for recharging (Figs. 1 and 2).

The second installation, replacing a 4½-ton General Electric trolley-type, was made on No. 12 level, June 8, 1913, when a Westinghouse storage-battery type (Fig. 3) of the following specifications was put into service:

Weight	7,600 lb.
Full-load speed, per hour	3.6 miles
Running drawbar pull on clean dry rails	1,920 lb.
Starting drawbar pull with sand	2,400 lb.
Gauge	24 in.
Diameter of wheels	24 in.
Wheel base	3½ ft.
Height over all, exclusive of trolley	3 ft. 10 in.
Length over all, exclusive of bumper blocks	9 ft. 6 in.
Motors	2

Battery, Edison type "A 8," consisting of 70 cells, with a rated discharge capacity of 300 amperes.

The third installation was made May 11, 1914, on No. 13 level, with a General Electric storage-battery locomotive of the following specifications:

Weight, including battery, approximately	8,000 lb.
Rated drawbar pull on level track	2,000 lb.
Speed at rated D.B.P., per hour	3 miles
Voltage	72
Gauge	24 in.
Overall length, not including couplers	10 ft. 3 in
Overall width	42 in.
Height over platform	29¼ in.
Height over battery compartment	46 in.
Wheel base	38 in.
Diameter of wheels	20 in.
Motors	2

Battery consists of 70 Edison type "A 8" cells having a rated discharge capacity of 300 amperes. The line voltage necessary for full charge must be at least 130 volts D. C.

Each of the last two locomotives has only a single battery of cells, of sufficient capacity to run on full load for the required length of time.

Charging

For charging purposes, a motor-generator set, 125-volt, three-phase, 50-h.p. motor, located in the hoist room on No. 9 level, takes current from the 2,300-volt A. C. line and delivers 125 volts, D. C., to the charging stations on each level. It was designed for the ultimate capacity of storage-battery locomotives on all levels from 10 to 14 inclusive.

A wattmeter is cut into the line at the motor-generator set, recording the amount of current delivered to it from the 2,300-volt A.C. line. Of course, there are losses, which occur in the motor-generator set, in the transmission of the current from it to the batteries, and from the batteries to the motors; but the total kilowatt-hours consumed is taken from the wattmeter, so the costs figured on that basis include these losses.

The motor-generator set is supplied with overload circuit breakers and reverse-current relay, which protect it from overload as well as from

the possible reversal of current due to low voltage or stopping of the motor generator while charging.

Rheostats are provided at each charging station so that both voltage and amperes can be delivered to the battery as required, as locomotives are sometimes called upon for extra heavy duty during one shift and the charging current during the interval between shifts is often made at twice the normal rate.

After some practice it was found that the two locomotives, supplied with a single set of batteries each, would operate successfully for two shifts on the following charging schedule and not give any signs of weak batteries.

Schedule for Charging—Starting at 8:00 *a.m.*

Operate without charging	4 hr.
Charge	30 min.
Operate without charging	4 hr.
Charge	2 hr.
Operate without charging	8 hr.
Charge	5 hr.
Idle without charging	30 min.
Total	24 hr.

Performance

It is not the intention here to make any comparison of the merits of the different types of storage-battery locomotives, but simply to give facts, as we have been able to get them, on the performance of these locomotives under the working conditions as they are here. Two of the locomotives, the Jeffrey on No. 11 level and the Westinghouse on No. 12 level, haul trains made up of seven and nine cars respectively, of 34 cu. ft. capacity each, or approximately 5,000 lb. of ore per car, on a track whose average grade is one-half of 1 per cent. in favor of the load, and are working full time. The General Electric on No. 13 level has only a light tonnage to handle and consequently does not work full time. This level has only been opened up recently and has not reached a normal output. But in collecting data for this article, it was impossible to separate the current consumption for the three locomotives, so that these costs are figured collectively for the three, with the average tonnages and distances hauled for each level figured separately and then averaged for the three. As will be seen, the costs for the General Electric locomotive, in kilowatt-hours consumed per ton of ore hauled, are much higher than for the other two. This is due to the fact that part of the ore on that level is trammed by the shovelers, making it impossible to get exact tonnages hauled by the locomotive. Also, most of the work of this locomotive has been in hauling waste, of which no account has been kept, so that the actual tonnage hauled is much greater than that stated.

The average kilowatt-hours delivered through the motor-generator set to the charging circuit for all three locomotives, taken from the watt-meter readings over a period of seven months, is 6,845 kw-hr. per month, and the total average ore tonnage hauled by the three locomotives is 27,660 tons per month, which gives a kilowatt-hour consumption of 0.2474 kw-hr. per ton of ore hauled.

The average distance that all three locomotives pull a loaded train is 730 ft., which gives 0.0339 kw-hr. consumed per ton of ore hauled per 100 ft., or for the round trip, $\frac{0.0339}{2} = 0.01695$ kw-hr. per ton per each 100 ft. the train is moved.

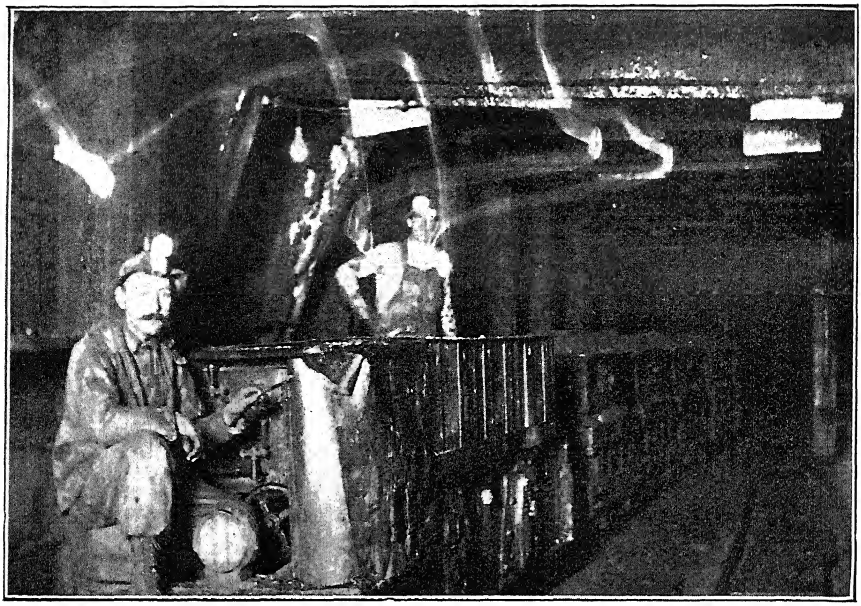


Fig. 3.—4-ton Westinghouse Storage-Battery Locomotive on No. 12 Level, showing the Type of Car Used for Haulage Purposes.

These figures are compiled on the average tonnages taken over a period of five months, but do not represent the full duty of the locomotives, since each level has a considerable tonnage of waste rock to be transferred every day, and consequently the kilowatt-hour consumption per ton of all material moved is appreciably less.

Efficiency

As has been said, losses occur in the transmission of current from the A. C. lines to the motor-generator set, and from the generator set to the batteries, and again in the transmission from the batteries to the motors. The normal voltage of the cells is approximately 110 volts, but, owing to a

peculiarity, characteristic of the Edison cell, drops to about 75 volts when the load is applied to the motor.

The efficiency and drawbar pull of mining locomotives are usually calculated by the manufacturer on the basis of clean dry rails and true roadbed; but this condition does not prevail in most mines, and especially in the Bunker Hill & Sullivan, where the tracks are often on the sill floors of stopes. For this reason, the roadbed is rough and irregular and has many short curves, and, moreover, overhanging chutes are continuously dropping small rock and water directly on the rails, which has to be crushed by the car wheels on each return trip and thereby greatly increases the normal drawbar pull required.

Maintenance and Repair

Since the installation of the storage-battery locomotives, the repairs on the motors and batteries have been practically nothing. The low voltage, as compared to the 500-volt D. C. used on the trolley-type locomotives, practically eliminates brush and commutator troubles, which always have been a source of heavy expense. About the only charge against the batteries is the time of one man for a few minutes each morning, giving them the daily inspection and refilling the cells with distilled water to replace that evaporated during the previous day—the amount of distilled water required for three batteries being about 20 gal. per week. In addition, there is a monthly charge, not exceeding $10 per battery, for cleaning and general overhauling. The principal source of repair expense on the locomotives to date has been new wheels.

The figures given below are the total average monthly cost of repair and upkeep from the date of installation to Nov. 1, 1914, and also include the cost of installation, which was quite large, because the battery boxes had to be altered and partly rebuilt, to adapt them to our charging system, and to protect them from the water issuing from the chutes under which they pass.

Storage-Battery Locomotive

	Average Monthly Repair Cost	Average Monthly Tonnage Hauled	Repair Cost Per Ton
2½-ton Jeffrey, No. 11 level	$48.513	13,501	$0.00359
4-ton Westinghouse, No. 12 level...	55.456	12,755	0.00434
4-ton Gen. Electric, No. 13 level ...	28.612	2,406	0.01189

The last figure is high because the costs are figured only on the tonnages of ore hauled, and most of the material hauled by this locomotive has been waste.

A comparison of these costs with those of the trolley locomotives which they replaced may be interesting. They cover a period of two

months in the first case and four months in the second, so that the figures must not be taken to represent an average cost over a long period. Separate repair costs for all locomotives were not kept until January, 1913, which accounts for the short period taken for the above locomotives, when it is remembered that they were replaced by storage-battery locomotives in March and June, respectively, of the same year.

Trolley Locomotive

	Average Monthly Repair Cost	Average Monthly Tonnage Hauled	Repair Cost per Ton
2½-ton Jeffrey, No. 11 level.......	$39.88	9,154	$0.00435
4½-ton Gen. Electric, No. 12 level..	93.912	14,645	0.00641

These figures do not include the initial cost and upkeep of the trolley wires and track bonding, which kept two men busy practically all the time, and which consequently was a heavy expense. No separate costs were kept for these levels, however, so they must be omitted in this connection.

It has been estimated by the company's electrical engineers that, with a few minor improvements in the charging system, and a better understanding of, and more careful attention given to, the operation of these locomotives by the motormen, the costs of repair and operation will be 75 per cent. less than for the trolley locomotives doing the same work.

Advantages of Storage-Battery Haulage

Different conditions in different mines make the haulage problem one to be worked out to suit individual needs; yet some of the advantages of storage-battery haulage will apply to all; and others, as we have found them here, may be of interest to those who are contemplating the installation of a haulage system.

First of all, storage-battery haulage does away with the dangerous trolley wire. Many of our drifts are on the sill floors of the stopes, where the timbers are depressed and often broken by the weight of the filling above, so that there is always danger of the employees coming in contact with the low-hanging, high-voltage trolley wire. This is not only dangerous to the train loaders and the repair men in the drift, but equally so to the miner walking through the drift, or climbing up a manway with his tools and drill steel. During the 17 years that the mine has been equipped with electric haulage, there have been three fatal accidents caused by contact with the trolley wires.

Another important feature is the easy access to the face of a drift with the storage-battery locomotive. There is no trolley wire to blast

against and break, with the attendant delay of finding the electrician to make the necessary repairs; neither is it necessary to wait for him to extend the trolley wires and track bonding as the face advances.

As mentioned in a previous paragraph, the cost of installation and upkeep of trolley wires and track bonding is a big item; and additional to that is the cost of replacing broken trolley poles and trolley heads, and the delay occasioned thereby.

Trolley-locomotive haulage necessitates provision for the return of the grounded current. Here that is done by a wire connection between the rail of the drift and a rail in the shaft. This connection has become broken several times, and consequently the current has been conducted by means of the wet board flooring to the iron pipes of the pumping system. In two instances the resulting electrolysis made a hole through the intake pipe of the pump, preventing it from drawing water.

These are only a few of the more important advantages of storage-battery haulage in underground practice, and there are many minor ones that could be added; but they may all be summed up in three words —safety, economy, and efficiency.

The management and the electrical staff are thoroughly satisfied with the results obtained from the installation, and are firmly convinced that with the care of the batteries better systematized and in the hands of competent, conscientious men, the ultimate results will exceed their original expectations.

Acknowledgments are due to W. C. Clark, Electrical Engineer, and M. J. Bottinelli, Assistant Electrical Engineer, of the Bunker Hill & Sullivan Mining & Concentrating Co., for their assistance in collecting data for this paper.

Discussion

Girard B. Rosenblatt, Salt Lake City, Utah (communication to the Secretary*).—The storage-battery locomotive certainly has its place in metal mining and I am glad to note that it is now receiving the recognition that is its due. It is, however, not a panacea for all conditions of mine haulage. There are places where it will not work to advantage and cannot compare with the trolley-type locomotive in cost of operation or in cost of maintenance.

On the one hand, there are places where it clearly outshines the trolley-type locomotive. Much depends on the headroom available in the drifts and on the condition of the roof. For short hauls with moderate-weight trains, where the roof is low and the ground heavy, nothing can compare with the storage-battery locomotive for convenience, flexibility, and above all, ability to stay on the job without the loss of time required

* Received Jan. 26, 1915.

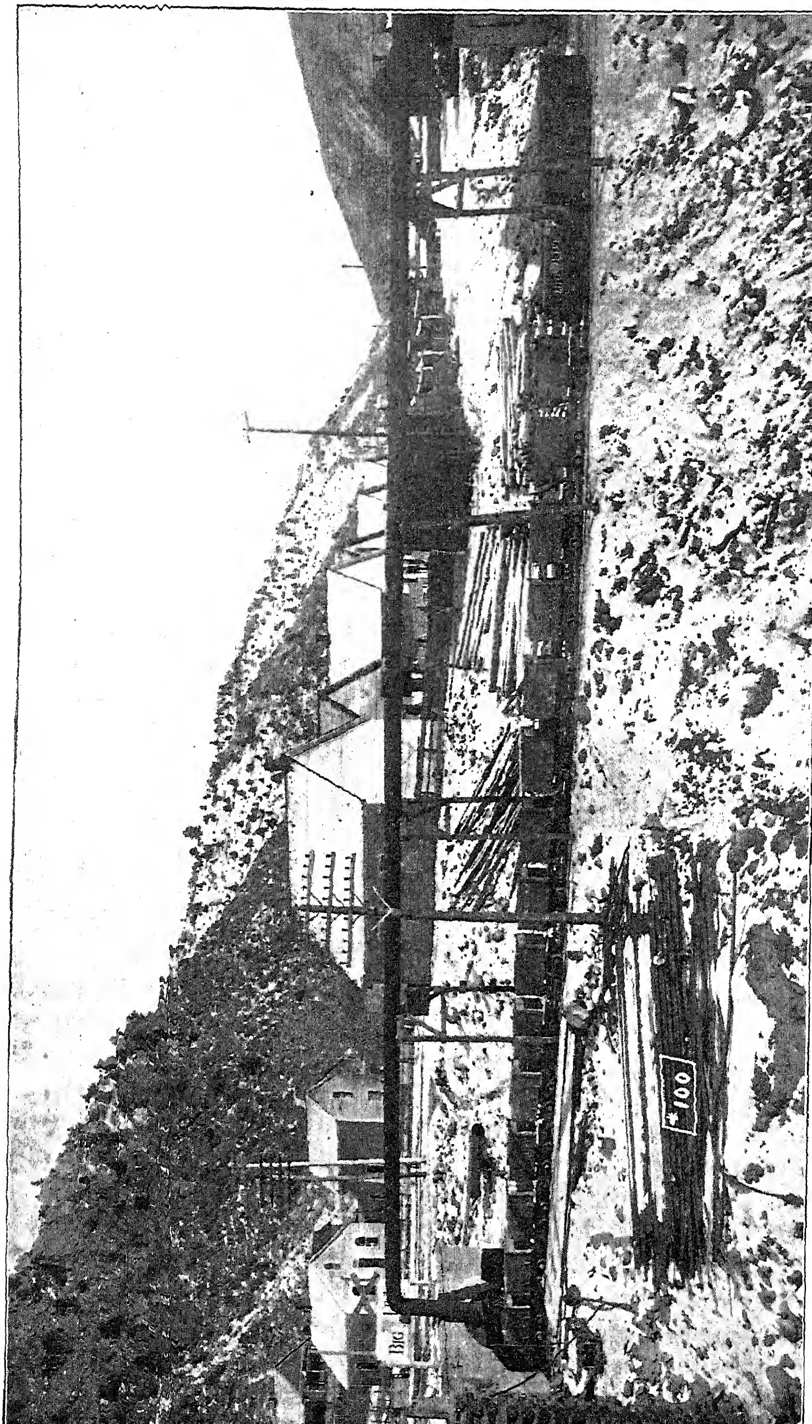


FIG. 4.—HAULAGE AT THE BIG FIVE MINING CO. BATTERY CARRIED ON A TENDER WHICH IS SEEN AHEAD OF THE LOCOMOTIVE.

to fix the trolley wire, to replace trolleys, etc. The power economy of a storage-battery locomotive is not as good as that of a trolley locomotive doing the same work. But no mining man cares for a few kilowatt-hours more or less if the use of them can obviate delays in getting out ore.

On the other hand, for long hauls with heavy trains, the storage-battery locomotive cannot compete successfully with the trolley type. The nearest approach to the successful commercial operation of a storage-battery locomotive on long-haul work is that at the Big Five tunnel in Colorado, where 80-ton trains are handled, and in this installation the size of battery required is so large that it cannot be carried on the locomotive, but is mounted on a tender, as shown in Fig. 4.

Records at the Big Five covering a period of six months show a monthly repair charge on the 4-ton Westinghouse storage-battery locomotive with its tender of $12.60 per month, and a repair charge per ton-mile of approximately $0.008. It may be interesting to compare these figures with figures that may be forthcoming from other sources. In analyzing these costs, the high cost per ton-mile over the six-month period is due to the fact that for three months the tonnage haul was very light indeed and the figure given above is the average over six months. For two months during which normal tonnage was handled, the repair cost per ton-mile was about $0.0065, or approximately the same figure as arrived at by Mr. Gwinn for operation on the No. 12 level.

Referring to the installation at the Bunker Hill and Sullivan, it may be of interest to know that the Westinghouse locomotive reported upon is not of a standard storage-battery type, inasmuch as the manufacturer was required to furnish a locomotive suitable for both trolley and storage-battery operation. This necessitated a locomotive with a somewhat different arrangement of controller and wiring than would be required for straight storage-battery operation, and required more complications than are included with the standard storage-battery locomotive. Still the operation of this locomotive has been satisfactory, though its efficiency when operating off the storage battery is not as high as would be the case with a straight storage-battery locomotive. The locomotive does include the special form of control furnished by the Westinghouse company on most of its storage-battery locomotives whereby high draw-bar pulls at slow speeds may be obtained with a less draft of current from the battery than is possible with the usual type of control furnished on trolley-type locomotives. The locomotive with the trolley pole attached is shown in Fig. 5.

It would appear that the average cost of repairs given by Mr. Gwinn in the table on the seventh page of his paper is not a direct indication of the mechanical cost of operating the locomotives because, as the author states, certain changes in the construction of the locomotives made by the mining company before the locomotives were put into operation

seem to be pro-rated over the time of operation. Therefore, we are apt to get the curious result that the longer the locomotive remains in operation the lower will be the monthly repair cost charged against it.

It would also seem in comparing the repair costs on the storage-battery locomotives and on the trolley locomotives given by Mr. Gwinn that conditions differing on the various levels have something to do with the repair costs. Combining Mr. Gwinn's two repair-cost tables on

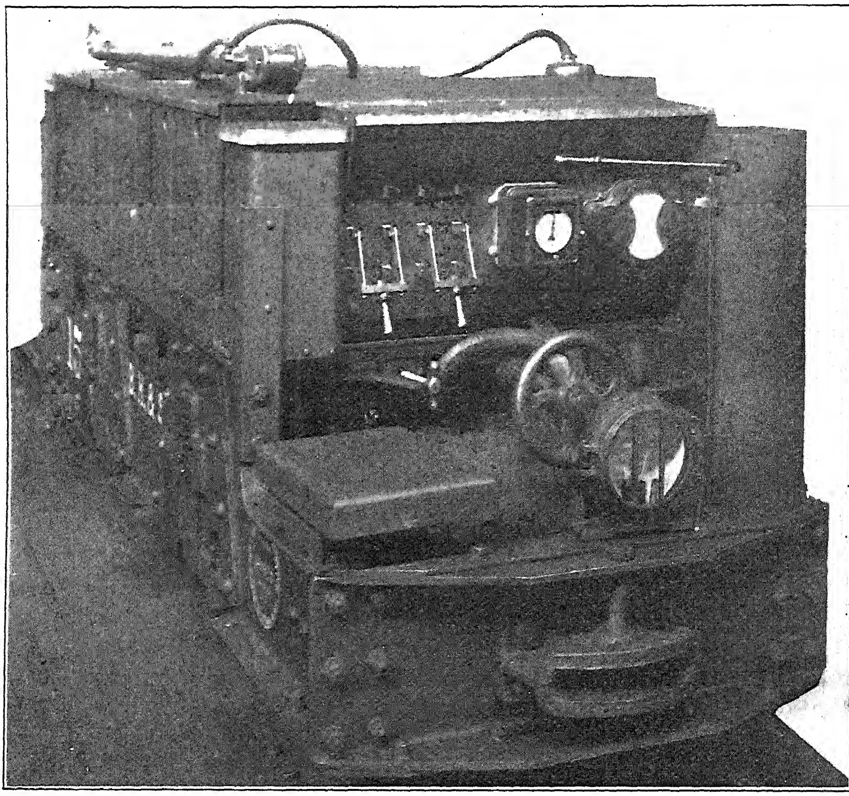


Fig. 5.—Westinghouse Storage-Battery Locomotive at Bunker Hill & Sullivan. Note Trolley Pole and Connection. Also Switch for throwing motors on to either Trolley or Battery.

storage-battery and trolley locomotives, respectively, into one table (see p. 235) makes it seem that conditions on the No. 12 level are more arduous than on the No. 11 level. It would be interesting to hear from Mr. Gwinn how these conditions actually differ.

The higher maintenance cost of the trolley locomotives compared with the storage-battery locomotives is certainly contrary to the usual belief of operators, and it is interesting to have such exact figures based on carefully kept records. The keeping of records which will permit the

	Type of Locomotive	Monthly Repair Charges	Repair Charge per Ton	Reduction in Costs Obtained by Using Storage-Battery Type, Per Cent.
Level No. 11	Storage battery	$48.513	$0.00359	17.5
	Trolley	39.88	0.00435	
Level No. 12	Storage battery	55.456	0.00434	32.2
	Trolley	93.912	0.00641	

maintenance cost of locomotives being calculated per ton-mile is certainly a distinct advance in scientific mine management. I do not know of any metal-mining property where such records are available except possibly the Phelps-Dodge properties in Arizona. Experience at other mines where both trolley and storage-battery locomotives have been operated under approximately similar conditions has led to a belief that the maintenance cost of storage-battery locomotives per ton-mile is higher than similar maintenance costs of trolley locomotives. There have, however, been too few actual records kept for any one to make any general positive statements regarding this matter. The beliefs in vogue are based on definite impressions rather than upon definite facts. At one property in Montana it was found that the principal cost of maintaining a storage-battery locomotive was the necessity of periodic semi-annual overhauling of the storage-battery cells and the battery boxes. A certain mine in Alaska, which operates several storage-battery locomotives, found periodic overhauling and painting of the battery box necessary to maintain continuity of service. Most mines operating storage-battery locomotives have had some trouble in repairing damage to the charging devices, particularly the plugs and plugging receptacles. None of these detailed troubles have been mentioned by Mr. Gwinn, and it would be interesting to hear whether they have been experienced at the Bunker Hill & Sullivan.

That storage-battery locomotive haulage can be thoroughly successful and satisfactory, and is so at the Bunker Hill & Sullivan, is borne out by the fact that affiliated properties in Alaska, the Alaska-Juneau Gold Mining Co. and the Alaska-Treadwell Gold Mining Co., have both installed storage-battery locomotives on the basis of the experience with this type of haulage at the Bunker Hill & Sullivan. The Alaska-Juneau Gold Mining Co. has within the last year placed in operation one 5½-ton Westinghouse-Baldwin storage-battery locomotive, and the Alaska-Treadwell has in the same period installed

two 4½-ton Westinghouse-Baldwin storage-battery locomotives. One of these is illustrated in Fig. 6.

The successful use of storage-battery locomotives in underground workings necessitates the realization that their inherent electrical characteristics are different from those of the trolley locomotive. The storage-battery locomotive carries its own power plant with it, and that power plant has a very decided characteristic in that the pressure under which it delivers its power falls off very rapidly when the power is with-

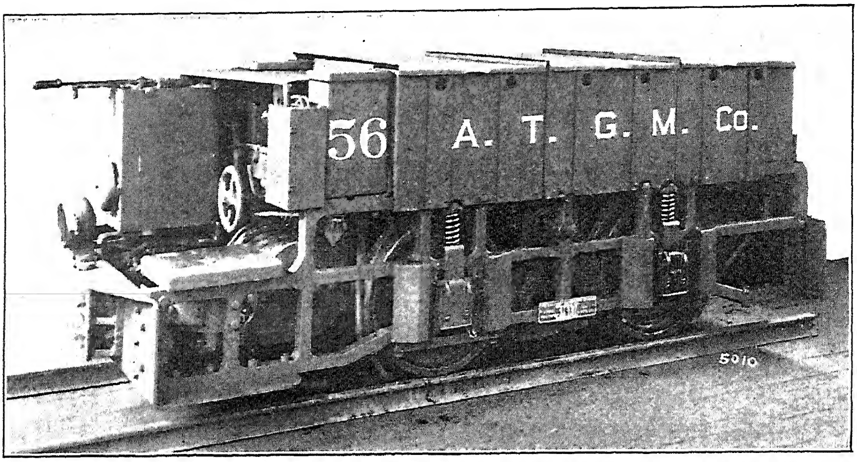


Fig. 6.—4½-Ton Storage-Battery Locomotive for the Alaska Gold Mining Co. Note the Bar Steel Frame.

drawn at a high rate. In some respects it is like a small steam boiler, the pressure of which will fall off rapidly if too great a draft of steam is required. The operator who would successfully use storage-battery locomotive haulage should bear in mind the following:

Do not try to use too small a battery; a safety factor in battery size will pay for the increased cost of battery within a year.

Do not use motors too big for the battery; the motorman will get all he can out of the locomotive and big motors give him an opportunity to club the battery.

Do not get a locomotive mechanically unsuited for mine service; underground conditions are quite different from surface conditions, and a locomotive to be successful underground must be designed and built as a mining locomotive.

John Langton, New York, N. Y.—I would like to call attention to p. 228, where the average kilowatt-hours per ton hauled are given. They are worked out per ton of ore hauled per 100 ft. Reducing to a ton-mile basis, it comes to 1.79 kw-hr. per ton-mile of ore. Compare the paper read a year ago by C. Legrand on storage-battery locomotives

in the mines at Bisbee. In that paper he gave certain figures which I will read, as follows: "The power required at the power station per useful ton-mile was approximately double that required with trolley locomotives, or 1.6 kw-hr.," so we have 1.6 kw-hr. to compare with the figure of 1.79 kw-hr. given in this paper. These are in substantial agreement, and constitute useful data for estimating. But although the amount of power taken, as Mr. Legrand states, is about twice that of the trolley locomotive, still the additional cost of 0.8 to 1.0 kw-hr. per ton-mile is such a small part of the total cost of hauling that it will seldom need to be considered if other advantages gained by the use of storage-battery locomotives should make them preferable to a trolley system.

H. M. Crankshaw, Lansford, Pa.—Mr. Gwinn states that the original cost of the storage-battery locomotive is high. We should like to know how much higher it is than the ordinary trolley locomotive. Then another feature which will have to be taken up will be the life of the battery. How long will the battery last, and what is the cost of replacing the battery? It seems to me that depreciation might play a very important part in the monthly upkeep of this locomotive if it were properly reckoned in.

Another point—Is the storage-battery locomotive safe? I do not mean safe as compared to the trolley locomotive, but could it be constructed so as to work in a gaseous place where, for example, mules work to-day? If it could, then it might be a useful help in haulage, otherwise I do not see that it is much of a help.

H. H. Clark, Pittsburgh, Pa.—The Bureau of Mines is greatly interested in storage-battery locomotives because, as Mr. Gwinn has pointed out, the use of storage-battery locomotives will eliminate the trolley wire, which is about the most dangerous piece of electrical equipment used in mines.

We therefore welcome any report which tends to establish the practicability and economy of storage-battery locomotives.

The impression that I have received from a rather hasty reading of Mr. Gwinn's paper is, however, almost too good to be true. The paper seems to state, without qualifications as to size or character of service, that storage-battery locomotives have advantages over trolley locomotives with respect to economy and efficiency as well as in respect to safety.

While it seems reasonable to believe that storage-battery locomotives can be maintained as cheaply as trolley locomotives, the question arises, Will the operating costs and the charges for interest and depreciation of the storage-battery locomotive and its charging equipment compare favorably with those of the trolley locomotive? And will this be true for all classes of service; that is, for long hauls and heavy hauls as well as

for short hauls and lighter loads? Or is the superiority of the storage-battery locomotive limited to certain kinds and conditions of service?

George R. Wood, Philadelphia, Pa.—I can possibly contribute a little practical experience to this discussion. We installed about nine months ago two 7-ton storage-battery locomotives in a coal mine, for reasons of safety. These locomotives are intended to be used on room entries and in pillar work where, under the Pennsylvania mine law, we are not allowed to use trolley locomotives. So far as the State Department of Mines is concerned, they are acceptable, although they would not be considered safe for working in gas. The motors, of course, are inclosed, as ordinary mining motors are, but to no greater extent, but the sparking, if any, would be at a much lower level than on the trolley, and if the gas got so heavy as to reach down that far the mines would not be safe to work under any circumstances. These particular locomotives are operating under probably the most adverse conditions of any locomotives in the country to-day, and among other things they are taking the empty cars up a 14 per cent. grade.

Our first experience was that one of the locomotives was smashed almost completely by running away going down loaded. That was the fault of the mechanical construction, of course, and was due to the fact that owing to the weight of the battery the mechanical part was too light. In other words, in the case of a 7-ton locomotive there were 4 tons of battery and only 3 tons of locomotive, and that required certain forms of construction, and we got it too light.

Since being repaired, these locomotives have been operating practically every day, handling from 80 to 95 wagons per day, gathering them from the rooms, collecting them, and delivering them to the trolley locomotive on the cross heading.

We have not as yet kept any figures of cost, because the repairs have been trifling; in fact, there have been no repairs to the battery except one set of lead connections which were burnt out by a short-circuit. The cost of these locomotives was almost exactly twice that of trolley locomotives for the same capacity and same work, and the work they will do depends almost entirely on the facilities for charging. These locomotives are charged at night, 7 hr. each, and we take an hour's boosting at noon. Without that hour's boosting it would reduce the capacity about 20 per cent. We are now arranging to charge the locomotive right at the work, at the point where it delivers the load to the trolley locomotive, which will enable us to charge from the trolley through resistance during the 5, 10, or 15 min. periods the locomotive is waiting for the cars. We expect to get ultimately to the capacity of the trolley locomotive, which we have not been able to do yet.

I do not believe that the storage-battery locomotive will be an

economical proposition compared to the trolley locomotive in coal mines under ordinary conditions. I think it will be very useful for advance work, and where there is a small amount of work on a long heading, or possibly in opening the mine. I believe that where it is required on account of safety, it will require a considerable amount of development before it will pass the requirements of the Bureau of Mines as a safe electrical proposition. We consider their requirements for mining machines to be very severe; not only must the motors be flame proof, but all the controlling apparatus, fuses, and things of that sort must be either operated under oil or very completely inclosed.

I may say that the cost of these locomotives was $4,000 each, of which the battery was exactly half, and the battery we used is the cheaper of the two batteries in common use. The other one would cost $1,000 more for the same capacity.

The Testing and Application of Hammer Drills

BY BENJAMIN F. TILLSON, FRANKLIN FURNACE, N. J.

(New York Meeting, February, 1915)

THE hammer drill rightly receives the credit for having made the one-man drill possible, and so many economies seem possible through the proper application of different types of hammer drills to various mining, quarrying, and excavating operations, that an indication of the economies effected by the New Jersey Zinc Co. at its Franklin mines may be of pertinent interest. When this company commenced its trials of hammer drills in 1907, these tools had not been developed to one-fourth the capacity and refinement which they have at present. At that time it was frequently stated that such a small tool, drilling holes of less diameter than the reciprocating rock drill, could not drill enough holes in a shift to permit the placing of sufficient explosive to break a tonnage of ore comparable with that produced by the "rock drills;" that the placing of small holes inclined upward, at angles steeper than 40° above the horizontal, could not be expected to produce results equal to the large flat, wet or dry, holes in the breasted back of an overhand stope, and would only shatter the ground so as to make the back unsafe. In spite of these adverse opinions, the hammer drills first showed their superiority over both heavy and light reciprocating drills in raising and in stoping, and then in drifting and quarry work. As a result, all of the reciprocating drills at the Franklin mines were scrapped three years ago, all of the mining work being accomplished with increased efficiency, as shown in detail in this article.

With the advent of the hammer drill in this property, it was considered advisable to make comparative tests of all the tools accessible, and it has since been the policy to investigate the merits of any advance of the drilling art in order to get the maximum amount of work from the tools. The necessity of devising some means of standardizing drill tests, and of measuring the consumption of compressed air as well as the drilling speed, was early realized.

The common test was to fill a measured air receiver with compressed air at a certain gauge pressure, run the drill until the pressure had dropped to too low a figure, then compute from the time, drop in pressure, and

capacity of the receiver, the cubic feet of free air used. This was not considered a fair indication of the drilling capacity of a machine, since the performance of some drills did not vary directly with the absolute pressures of the compressed air.

It was, therefore, found expedient to build a water-displacement air meter with which the drill test could be carried on for any length of time without serious variations in the desired air pressure. This apparatus, as shown in Fig. 1, consists of two tanks, half-filled with water, and made of 12-in. pipe with blank flanges, gauge glasses being mounted on one, a four-way cock connecting the compressed-air supply pipe with both tanks and the air line going to the drill. This device gives more accurate results than the common types of water or gas meters; and since any errors are due to the human element of reading the gauge glasses and reversing the four-way cock, they tend to be compensating throughout a number of tests.

The procedure is as follows: Air is drawn by the drill from the receiver *C*, which tends to trap any moisture carried over by the air from tanks *A* and *B* and assures a constant pressure while the four-way cock is reversed. In the arrangement shown in Fig. 1, the receiver draws its supply of air from the tank *B* and the water rises in this tank by virtue of the pressure of the air admitted from the air main through the four-way cock to the top of tank *A*, where the water is being forced downward and through the 2-in. connecting pipe to tank *B*. When the water has risen to a certain point near the top of the gauge glass in tank *B*, the four-way cock is reversed and the inlet air is supplied to the top of tank *B*; the drilling air is then taken from the top of *A*, the reversal of the cock again being made when the water in tank *B* has fallen to a point near the bottom of the gauge glass.

A pet cock is placed on the top of tank *C* so as to permit the bleeding of air to bring the water columns to the desired point for starting a run, and another pet cock is attached to the bottom of the same tank in order to permit the drainage of water. For convenience in measuring and computing, a run is made on the supply of compressed air indicated by a certain number of reciprocations of the water columns between fixed points on the gauge glasses; the pressures are measured on the air gauge mounted on tank *C*, the length of time is taken by a stop watch, and the consumption of free air per minute is computed. Unless a pressure regulator is installed between the four-way cock and the receiver *C*, or else a globe valve at this point is operated manually to throttle the air so as to maintain a constant pressure, it is evident that the air pressure at the drill will vary in accordance with the water column supported by the inlet air pressure, but since the gauge-glass marks are in this instance set 34¾ in. apart, the maximum variation in pressures is about 1¼ lb. per square inch; it is difficult to find a pressure regulator which will control a pressure of 90 or 100 lb. per square inch to a closer

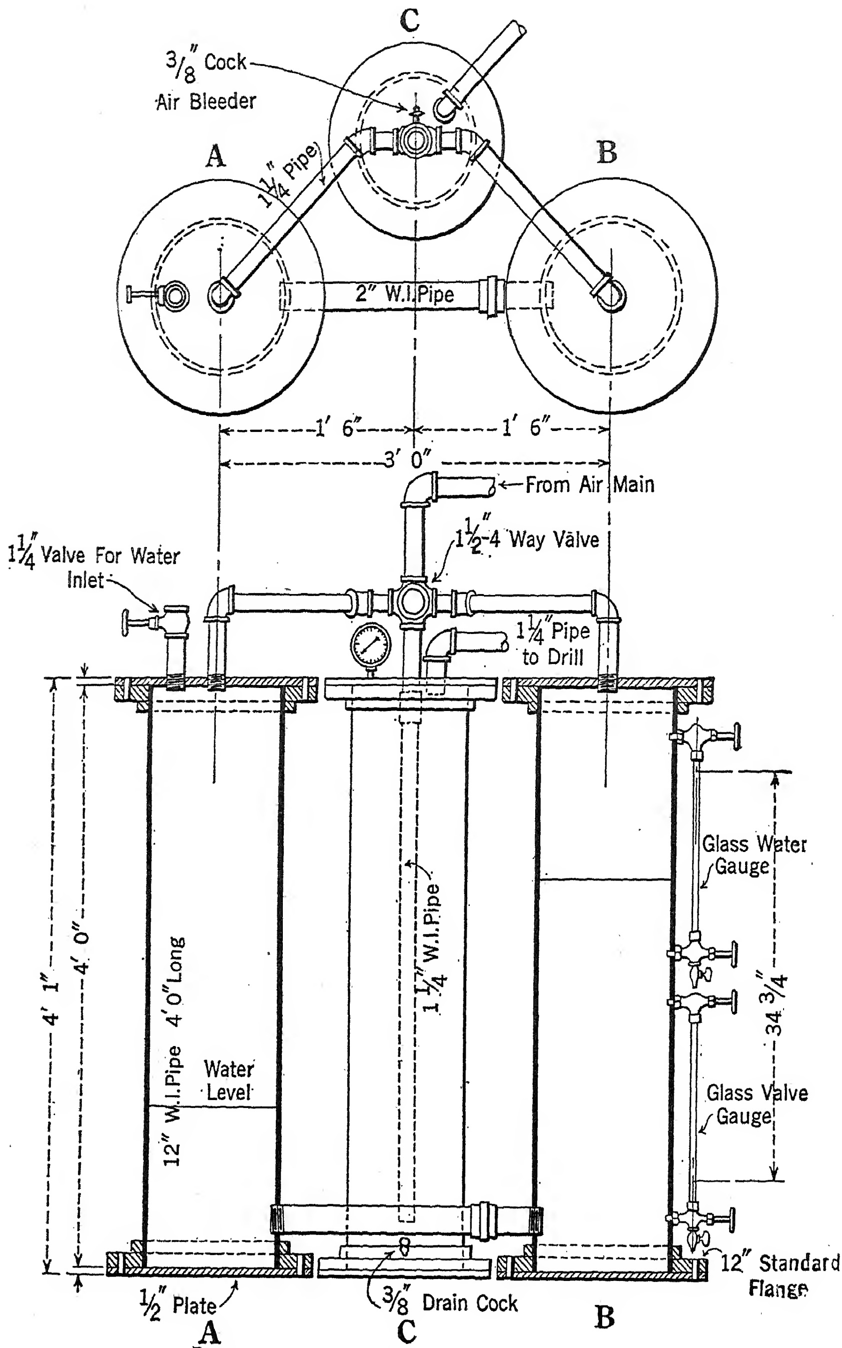


FIG. 1.—WATER DISPLACEMENT AIR METER FOR ROCK-DRILL TESTING.

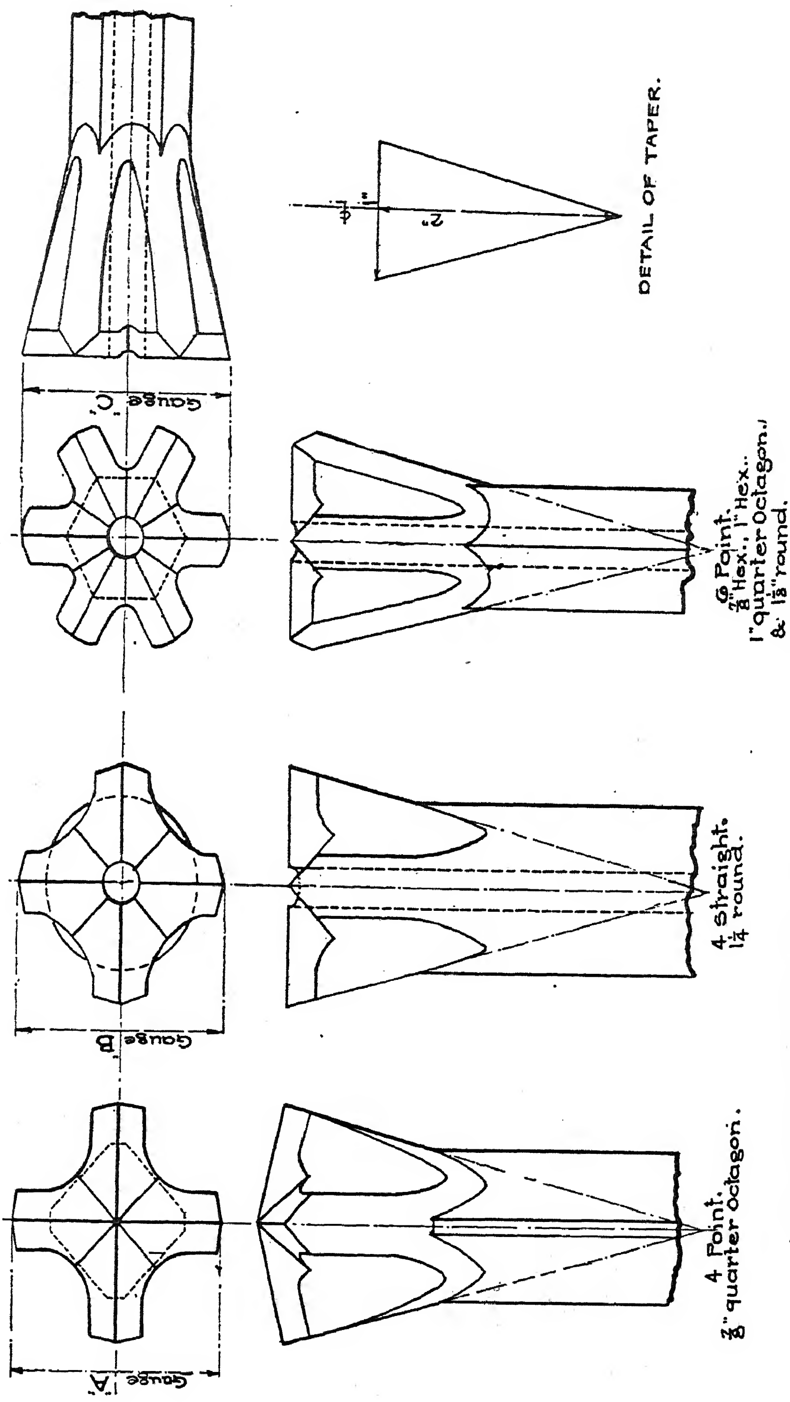


Fig. 2.—Types of Drill Bits used at Franklin Mines of the New Jersey Zinc Co.

degree of accuracy, and the sensitiveness of drills to air pressures and the accuracy of time and distance measurements rarely exceed this error.

It is obvious that in comparative drill tests the following factors must be considered: The nature of the rock drilled; the gauge of the drill bits and their form and condition; the maintenance of equal compressed-air pressures; similar inclination and approximate depths of drill holes; equal vigor in the rotation of hand-rotated tools; and proper fit of the drill shanks in the chuck bushings, as well as their construction so that the blows are delivered on a plane surface of proper size at right angles to the axis of the drill steel and at the center of its shank end. In the tests summarized in Table I, 1¾ in. has been taken as the standard diametral gauge of the bit, since it is a dimension which averages the gauges of drill steels used in reciprocating rock drills and is fair in determining the performances of such tools; it also represents almost the largest gauge necessary in hammer-drill stopers or block-holers, so that equal or even better performances may be expected from them as a hole is deepened with the smaller gauges in a set of drill steel. At Franklin, the testing rock is a compact coarsely crystalline white limestone, which greatly resembles a marble, and this rock proves a fair average of the various qualities of ore met in the mining operations. Although it is not hard, for a well-tempered drill bit can drill 3 or 4 ft. of hole before its cutting edges are materially dulled, and although it seems to chip freely, yet it possesses a compactness and toughness which is likely to prove surprising to one who has not previously tested a drill in it. Tests with various machines in Franklin, and elsewhere, indicate that this white limestone does not cut quite so fast as a sharp drill can achieve in Cripple Creek granite; is about on a par with Barre granite; and cuts slightly faster than Quincy granite. The chief difference is that a good drill will cut this limestone as fast in the second or third minute of its run, while it would have been dulled by the first minute's run in the granites, causing its cutting speed to fall off materially in the second and third minutes. Raised-center cross bits are the standard type used with solid steel in these tests, and flat-faced six-point bits are generally used with the hollow steels, in general the cutting speed of these bits being about the same if the rotation of the drill steel is free. Fig. 2 shows the forms of the drill bits used at the Franklin mines.

The results in Table I are not complete to date, but indicate the improvements in hammer drills, from the operators' standpoint of efficiency, during four years of advancement in the art; and it may be interesting to note that, so far as we know, no drills of the present day surpass in drilling speed and low air consumption the best drills listed in this table, although several makes of hammer drills are on a par with them. In order to avoid invidious comparisons between the different makes of drills, symbols have been used to designate each certain make and design of drill. The following abbreviations have been used in this table:

TABLE I.—*Summary of Tests of Representative Drill Types Made by the New Jersey Zinc Co.*

Symbol for Drill	Date	Type of Drill				Air Press. Lb. per Sq. In. (Gauge)	Shape of Drill Bit	Free Air, Cu. Ft. per Min.	Drilling Speed, Inches per Min.	Free Air, per Inch Drilled	Factor, In. per Min. Divided by Free Air per Inch	Condition of Bit and Remarks
		Rotation	Control of Piston	Style of Feed	Weight							General.—Air measured by displacement of water. Rock was white crystalline limestone. Gauge of bits was $1\frac{11}{16}$ to $1\frac{3}{4}$ in.
A	7/27/09	Hand.	Valve.	Dir. air.	75	93	Raised crux.	60.4	3.85	15.62	0.246	1 corner broken, with extension.
A_1	7/27/09	Hand.	Valve.	Dir. air.	84	92	Raised crux.	73.2	3.83	19.15	0.200	With extension.
B	7/27/09	Hand.	Valve.	Dir. air.	76	90	Raised crux.	90.0	3.20	28.20	0.118	
1B	7/27/09	Hand.	Valve.	Dir. air.	76	72	Raised crux.	76.8	4.28	17.90	0.239	Different drill of above model.
C	7/27/09	Hand.	Valve.	Dir. air.	64	90	Raised crux.	63.0	5.10	12.37	0.414	
D	7/27/09	Hand.	Valve	Dir. air.	74	90	Raised crux.	92.7	4.50	20.60	0.218	
D	7/27/09	Hand.	Valve.	Dir. air.	74	85	Raised crux.	86.2	5.52	15.6	0.354	Bit broken.
E	7/27/09	Hand.	Valveless.	Dir. air.	84	87	Raised crux.	42.0	1.93	21.7	0.089	
F	7/27/09	Hand.	Valveless.	Rev. air.	68	95	Raised crux.	69.8	3.38	20.3	0.167	
G	10/1/09	Hand.	Valve.	Dir. air.	78	88	Raised crux.	75.8	5.65	13.4	0.421	
G_1	10/8/09	Hand.	Valve.	Dir. air.	73	92	Raised crux.	79.5	5.91	13.4	0.450	
G_2	10/25/09	Hand.	Valve.	Dir. air.	73	88	Raised crux.	79.5	6.00	13.2	0.455	
G_3	12/6/09	Hand.	Valve.	Dir. air.	78	96	Raised crux.	83.5	7.00	11.9	0.588	
H	10/20/09	Hand.	Valveless.	Rev. air.	56	95	Flat hex.	29.8	2.46	12.1	0.203	No tappet.
1H	10/21/10	Hand.	Valve.	Rev. air.	..	93	Crux.	66.7	2.10	31.8	0.066	
J	12/15/10	Hand.	Valve.	Rev. air.	..	94	Crux.	66.9	7.30	9.15	0.800	High run.
J	12/15/10	Hand.	Valve.	Rev. air.	..	94	Crux.	66.3	6.70	9.92	0.675	Average 8 runs.
J	2/25/11	Hand.	Valve.	Rev. air.	..	94	Crux.	68.8	6.25	11.01	0.567	Average 23 runs.
K	3/2/11	Hand.	Valve.	Rev. air.	..	92	Crux.	77.6	7.29	10.65	0.684	
K	3/7/11	Hand.	Valve.	Rev. air.	..	94	Crux.	75.4	6.90	10.90	0.633	Average 10 runs.
L	4/10/11	Hand.	Valve.	Rev. air.	..	97	Crux.	58.0	6.29	9.21	0.683	
L	4/10/11	Hand.	Valve.	Rev. air.	..	97	Crux.	58.5	6.19	9.45	0.655	Average 7 runs.
L	4/18/11	Hand.	Valve.	Rev. air.	..	93	Crux.	55.8	6.55	8.52	0.769	Average 7 runs.
M	4/21/11	Hand.	Valveless.	Rev. air.	..	95	Raised crux.	54.7	9.35	5.85	1.600	High run to $1\frac{11}{16}$-in. bit.
M_1	6/24/11	Hand.	Valveless.	Dir. air.	..	99	Raised crux.	58.5	11.54	5.07	2.280	Bit a little soft.
1M	6/24/11	Hand.	Valveless.	Dir. air.	..	99	Raised crux.	58.5	9.09	6.45	1.410	Average 3 other runs, same machine.
M_2	8/30/11	Hand.	Valveless.	Dir. air.	..	100	Raised crux.	65.0	10.75	6.04	1.780	
M_2	8/30/11	Hand.	Valveless.	Dir. air.	..	99	Raised crux.	62.9	9.56	6.59	1.450	Average 11 runs.
N	9/27/11	Hand.	Valve.	Dir. air.	..	99	Raised crux.	36.0	5.28	6.82	0.774	
N	9/27/11	Hand.	Valve.	Dir. air.	..	99	Raised crux.	39.2	4.63	8.59	0.540	Average 11 runs.
N_1	10/24/11	Hand.	Valve.	Dir. air.	65	96	Raised crux.	66.4	2.67	24.8	0.108	Average 14 runs.
J	2/3/12	Hand.	Valve.	Dir. air.	..	97	Raised crux.	71.2	6.06	11.76	0.515	Drill "J" after 1 year's service.
X M_2	2/20/12	Hand.	Valveless.	Dir. air.	..	99	Raised crux.	62.9	11.10	5.66	1.962	High run.
X M_3	2/20/12	Hand	Valveless.	Dir. air.	..	100	Raised crux.	63.5	10.48	6.06	1.739	Average 4 runs.
X O	3/5/12	Auto aux V.	Valveless.	Dir. air.	..	98	Flat hex.	79.1	10.34	7.65	1.350	Hollow bit, no air through it, high run.
X O_1	6/3/12	Auto aux V.	Valveless.	Dir. air.	87	100	Raised crux.	85.5	9.55	8.95	1.068	With pressure control on feed ½ fd., bit with soft center, No. 6.
X O_1	6/3/12	Auto aux V.	Valveless.	Dir. air.	87	100	Raised crux.	87.0	8.50	10.24	0.830	With pressure control on feed, fine bit, No. 12.
P	2/16/12	Hand.	Valve.	Dir. air.	..	100	Raised crux.	67.0	8.43	7.94	1.062	Hard to rotate.
P_1	5/13/13	Hand.	Valveless.	Dir. air.	86	96	Raised crux.	55.8	10.76	5.18	2.080	Rotates freely.
P_1	5/13/13	Hand.	Valveless.	Dir. air.	86	97	Raised crux.	59.5	11.03	5.40	2.043	Rotates freely.
P_1	5/13/13	Hand.	Valveless.	Dir. cir.	86	96	Raised crux.	56.8	9.32	6.10	1.529	Average 9 runs
P_1	5/13/13	Hand.	Valveless.	Dir. cir.	86	96	Flat crux.	57.1	7.98	7.17	1.110	Average 3 runs.
Q	5/16/13	Auto rifle.	Valve.	Rev. air	..	91	Raised crux	51.4	6.12	8.39	0.730	
Q	5/16/13	Auto rifle.	Valve.	Rev. air.	86	92	Db'le. chisel.	48.2	8.18	5.90	1.388	Hollow steel.
R	9/25/13	Auto rifle.	Valve.	Rev. air.	..	96	Raised crux.	112.0	6.85	16.25	0.421	Air bled to control feed press.
R	9/25/13	Auto rifle.	Valve.	Rev. air.	86	91	Flat hex.	102.2	7.50	13.63	0.550	Air bled to control feed press.

Auto aux V = automatic auxiliary valveless control of rotations.

Auto rifle = automatic rotation caused by a piston reciprocating as though it were controlled by a rifle bar.

Dir. air = direct-air feed, or one in which the feed cylinder is rigidly attached to the hammer cylinder and in which the feed piston or plunger extends from the rear end of the machine by virtue of the air pressure applied to it.

Rev. air = reversed-air feed, or one in which the feed piston is rigidly attached to the hammer cylinder and the feed cylinder is free to extend backward, so readily adapts itself to the customary column mounting of stoping drills.

Some tests were included in this table for the consideration of points to be made later. Before studying the improvements in hammer-drill efficiency it seems wise to explain the reasons for offering the figures in the last column of figures as representing a factor of "drill desirability."

In determining the relative merits of rock drills, whether of the reciprocating or hammer type, the logical basis is one of cost. Therefore, the drill which bores a foot of drill hole of standard cross-section at the lowest cost rate for drilling labor, power, and maintenance (including amortization), would have the highest "factor of desirability;" and a formula to express this may be developed as follows:

Let

F be the "factor of desirability,"
D be the cost of drilling labor per foot of hole,
P be the cost of power per foot of hole,
M be the cost of maintenance per foot of hole.

Then,

$$F = \frac{1}{D + P + M}$$

Let

t = Period of time for drilling-speed test, in minutes.
d = Depth of hole drilled in time, t, in inches.
$S = \frac{d}{t}$ = Drilling speed during actual running of machine, in inches per minute.
L = Hourly wage of drilling labor, in cents.
O = Percentage of time spent in drilling to total operating time including the changing of drill steels and shifting to new positions and starting of new holes.

Then

$$D = \frac{L}{\frac{60SO}{12}} = \frac{L}{\frac{5dO}{t}} = 0.2\,\frac{tL}{dO}$$

Let

p = Power cost to produce 100 cu. ft. of free air, compressed to standard drill-testing pressure, in cents.

v = Number of cubic feet of free air used in test by operating drill.

d = Depth of standard hole drilled, in inches.

Then

$$P = \frac{12\,pv}{100\,d} = 0.12\,\frac{pv}{d}$$

Substituting these values of D and P in the original equation we obtain

$$F = \frac{1}{0.20\frac{tL}{dO} + 0.12\frac{pv}{d} + M} = \frac{d}{\frac{0.2L}{O}t + 0.12\,pv + dM}$$

Since L is a constant for any particular mine, and O for a given number of steel changes with any particular type of drill—such as a column-mounted reciprocating drill, a column-mounted hammer drill, an air-feed hammer drill, a block-holing drill, etc.—we may simplify the equation by substituting

$$k = \text{coefficient of drilling} = \frac{0.20\,L}{O}$$

Also, since p is a constant for any particular mine we may further simplify by placing

$$k' = \text{coefficient of power} = 0.12\,p$$

and we then have the general equation for any particular mining conditions and type of drill

$$F = \frac{d}{kt + k'v + dM}$$

However, the correct value for maintenance and amortization of any particular type and make of drill can be determined only after operations extending over months or years, so that this factor may well be left out of a formula which is to be used for classifying drills after drilling speed and power consumption tests, which may be completed in a short time. The consideration of the reduction of drilling speed and the increase of power consumption, which occur in a drill because of wear or any other normal results of service, may fairly be placed in the same class as maintenance. Judgment as to the materials, workmanship, and design of any drill, as well as reports of its satisfactory service elsewhere, will lead to a rough estimate of the final desirability of a drill if it has shown a high standard, on testing based on drilling speed and power consumption.

The equation is thus simplified to the form

$$F = \frac{d}{kt + k'v}$$

But other highly important factors enter into the problem of selection of a drill, namely, the reduction in labor units, capital, and overhead charges brought about by an increased drilling speed and increased tonnage per machine; the increased efficiency of supervision and work caused by the reduction and concentration of the number of working places; the possibility of producing a greater tonnage from any property with a limited number of working places; and the possibility of reducing the drilling equipment, with its attendant stock of spares, hoses, and connections, and extensive air mains, if a drill with a greater drilling speed may be employed. It therefore seems that the following formula is more indicative of the actual merits of drills, although theoretically it has no derivation, and must be considered empirical; it also possesses the virtue of reducing to a simple form. This formula for a "factor of desirability" has been used for the past six years at the Franklin Furnace mines of the New Jersey Zinc Co. All coefficients have been omitted since the following drill tests have all been under the same standard conditions.

$$F' = \frac{1}{DPM}$$

Since M is treated separately, as has been previously suggested, the equation becomes

$$F' = \frac{1}{DP}$$

Now if the same values previously deduced for D and P are substituted,

$$F' = \frac{1}{\frac{kt}{d} \times \frac{k'v}{d}} = K\frac{d^2}{tv}$$

where K is a new coefficient equal to the reciprocal of the product of k and k'.

Therefore, the "factor of desirability" equals the drilling speed, in inches per minute, divided by the power consumption, in cubic feet of free air, per inch drilled. It is quite evident that the factor gained from the quotient of inches drilled per minute divided by cubic feet of free air per minute (or the reciprocal of this quotient) gives merely the power consumption per inch of hole drilled and ignores the quantity of drilling which may be accomplished.

The application of both of these formulas for F and F' to a hypothetical problem may be of interest to show the comparative results within the limits of practice.

Let us assume that 30 h.p. is required to compress 100 cu. ft. of free air per minute to 100 lb. per square inch gauge pressure and deliver the same to a drill in the mine; that the power cost is 1c. per horsepower-hour;

that a drill which shows a drilling speed of 10 in. per minute on test averages 20 ft. per hour under working conditions, and uses 60 cu. ft. of free air per minute on test; that another drill will show a drilling speed of 6 in. per minute on test with an air consumption of 36 cu. ft. per minute and will average 12 ft. per hour under working conditions, and that the wage scale for drill runners is 40c. per hour, then,

For the fast drill:

$$t = 1 \text{ min.}$$
$$d_1 = 10 \text{ in.}$$
$$L = 40\text{c.}$$
$$O_1 = 20 \div \frac{10 \times 60}{12} = 0.40$$
$$p = \frac{30 \times 1}{60} = 0.5\text{c.}$$
$$v_1 = 60$$

$$F_1 = \frac{d}{0.2\dfrac{L}{O_1}t + 0.12\,pv_1} = \frac{10}{\dfrac{0.2 \times 40 \times 1}{0.40} + 0.12 \times 0.5 \times 60} = 0.424$$

$$F'_1 = \frac{1}{\dfrac{0.20\,tL}{O_1d_1} \times \dfrac{0.12\,pv_1}{d_1}} = \frac{O_1d_1^2}{0.024\,tLpv_1}$$
$$= \frac{0.40 \times 100}{0.024 \times 1 \times 40 \times 0.5 \times 60} = 1.389$$

For the slow drill:

$$d_2 = 6 \text{ in.}$$
$$v_2 = 36 \text{ cu. ft.}$$
$$O_2 = 12 \div \frac{6 \times 60}{12} = 0.40$$

$$F_2 = \frac{6}{\dfrac{0.2 \times 40 \times 1}{0.40} + 0.12 \times 0.5 \times 36} = 0.271$$

$$F'_2 = \frac{0.40 \times 36}{0.024 \times 1 \times 40 \times 0.5 \times 36} = 0.833$$

Thus the relative factors for the two drills by the first formula have a ratio of 0.424 to 0.271 or 1.56 to 1; and by the second formula (empirical) the ratio of factors is 1.389 to 0.833 or 1.67 to 1. In other words, by the empirical formula the fast drill is credited with about a 7 per cent. higher rating than by the theoretical formula, and this does not seem an undue allowance to cover the unestimated advantages previously enumerated.

Records made previous to July, 1909, have not been shown in Table I since much of the work done in 1907 and 1908 was distinctly experimental in determining the desirable cylinder diameters, lengths of strokes,

piston weights, valve weights, etc., but such records show drilling speeds of about 2 to 3 in. per minute with air consumptions of from 40 to 70 cu. ft. of free air per minute at 90 lb. per square inch gauge pressure. The listed tests made during 1909 cover most of the well-known American makes of hammer drills at that time, and if one excepts the drills denoted by symbols G, G_1, etc., since they were experimental tools, the design of which was developed by the New Jersey Zinc Co. at Franklin Furnace, N. J., it is noticeable that about 4½ and 5 in. were the highest drilling speeds obtainable at about 90 lb. pressure and with an air consumption of 60 to 90 cu. ft. of free air per minute; and for various drills the "factor" varied from 0.09 to 0.41. Those drills marked G, which were made exclusively for the New Jersey Zinc Co., increased the drilling speed about 40 per cent. above the best previous drill performances, and remained unequaled in drilling speed for a year and unsurpassed for about a year and a half. The fact that a number of these drills were included in the equipment at Franklin accounts for part of the increased stoping efficiency during the year 1911, as cited later. Although it was then the opinion of some unprejudiced persons, well versed in the drilling art, that such tools had reached their practical limit of drilling speed as well as the limit of strengths of materials, yet 18 months later a new type of drill was developed to achieve 20 per cent. more drilling with twice as good a factor, and a renewed equipment of these other drills again increased the mining efficiency. Again a period of 18 months sufficed for the production of a hammer drill which still further advanced the drilling speeds 20 per cent., and since the introduction of this drill we have been able to find several drills which surpassed it 10 to 20 per cent. in drilling speed.

In Table I some seemingly freak runs are noticeable, which are included to call attention to the variability of results in presumably standard testing. For instance, under drills D it appears that a bit with two wings broken will drill faster and at a lower air consumption per minute than can be attained with a perfect bit; and again, with drill M_1, a bit which has proved a little soft and battered drills one-fourth more per minute than bits in proper condition and with the same air consumption. Furthermore, the tests of one person indicate that, when the size and form of the drill bits are the same, faster drilling can be done with short steels than long ones, while another investigator shows a greater drilling speed with long steel than with short. The use of tappets or anvil blocks between the shanks of drill steels and pistons is generally estimated as causing a reduction of 20 to 30 per cent. in the drilling ability, but some tests do not confirm this and show even an increased cutting speed with the use of anvil blocks in a machine otherwise the same. With some drills the use of water to clean the cuttings from the hole seems to cause a cutting speed below that obtainable through the use of

compressed air for the same purpose, but in other instances the advantages are reversed. In short, there seem to be so many variables in the drilling problem as to warrant a 10 per cent. variation in the results of supposedly standard tests, and a number of runs should be made to gain a fair average; or strict judgment of machines should not be made within this limit.

Perhaps the consideration of the physical phenomena relating to the process of drilling may prove of interest and value. When rock is excavated by a drill bit three applications of forces seem to be involved—by abrasion, by crushing, and by severing or chipping. Although all of these must take place to a certain degree, the greatest amount of useful work is performed when the percentage of force applied to chip reaches a maximum. But in rock it appears that chips can be produced in radically different ways: first, by the severing of molecules, and second, by the reflex forces produced in an elastic medium. To illustrate this, consider the chipping of a comparatively inelastic substance such as lead. With a hammer and a chisel, whose axis is inclined considerably from the normal to the surface of a lead block, it is possible to sever the lead and roll up chips, but if the chisel is normal to the surface of a thick block only an indentation can be made and there probably will be a raised area about the indentation to accommodate a certain percentage of the displaced metal. On the other hand, with a highly elastic material, such as glass, the forces impressed by a normally positioned chisel will cause a compression of the molecules, whose elasticity will cause their expansion toward a free, unresisted surface. Since the greatest forces are developed at the surface, since the penetration of the chisel carries some forces to a depth below the surface; and since the chisel surface itself applies some forces at an angle to its axis and impedes the re-expansion of molecules to the space it occupies, therefore, the reflex forces produce more or less cone-shaped chips or flakes and leave a corresponding crater in the block of glass. Now, if the chisel is placed near the edge of a block of glass, the blow upon it will induce stresses to another free face and a correspondingly larger chip will be produced because of the tendency of the forces to seek relief in the shortest direction as well as because of the severing effect. The method of cutting of a drill bit is commonly shown as taking place in this last way with the progressive chipping of a series of benches or steps, but it is doubtful whether such a procedure exists, except in rare instances, for the speed and latitude of rotation between consecutive blows of the drill piston or hammer cannot be controlled with sufficient precision nor adjusted to the various rocks; and an inspection of the cuttings from a drill hole shows them to be flakes, or a crushed and abraded powder.

In the formation of these flaky chips there may be a limiting force of blow for each velocity of impression in order to gain the most useful

work (*i.e.*, in the production of flakes), for it appears that beyond certain limits the blows increase the percentage of crushed material and the drilling speed does not vary with the force applied, so that some heavy-hitting drills accomplish more in medium-soft ground when a portion of their blows are absorbed by a tappet at the shank end of the steel or by a cushion of water intervening between the bit and the rock. If the force of the blows was lessened by a reduction in air pressure the speed of the piston would be slowed up, and the drilling would suffer from the fewer number of blows per minute.

The transmission of the kinetic energy of the piston to the rock is also influenced by many factors. The blow may be delivered against the rock by the free drill steel which is driven forward through the intervening air or water by the impact of the piston and the velocity of the steel will depend upon the relative masses of the drill steel and piston, the velocity of the piston, and the coefficient of elasticity of the steel, in accordance with the well-known laws of mechanics which deal with elastic or partly elastic bodies and their impact. The drill steel in this way assumes the functions of a "jumper" drill which is driven against and rebounds from the rock at a high frequency, and its action is well seen in most all screw-feed hammer drills with the ringing or jingling of the steel in a drill hole. Another mode of force transmission is by compressional waves, traveling through the drill steel from the shank to the bit. This latter condition brings a cutting effect only when one end of the steel is tight against the rock, but then proves very efficient. Although the air-feed hammer drills usually chatter the steel against the rock, like a projectile shot from the chuck bushing by impacts of the piston, yet it seems possible to approximate the other working condition by designing the air feed so that the pressure is lowered as the piston is traveling on its back stroke (possibly by taking the supply air, for the back stroke, from the air feed) and so that the air-feed pressure builds up and forces the drill against the rock just before it is struck by the piston. The reversed-air feeds may sometimes approximate these conditions and then assist the machine to a higher drilling speed. If hammer drills were made so that the drill steels were always held firmly against the rock, when the piston strikes them, it seems unquestionable that the greatest efficiency of the blows of the piston would result, providing they were properly timed, for no energy would be lost by reason of the inertia of the drill steel, but only that due to heating, resulting from the imperfect elasticity of the metal. The question of the proper timing of the piston blows opens another phase of the matter, namely, the reaction of the rock upon the drill steel; and this effect is the more pronounced with harder rock. It tends to speed up the piston and is so noticeable in running a machine against a metal block as to invalidate, as too high, all air-consumption tests so conducted. The effect of these reactive vibrations upon the drill steel may prove very

marked and serious. Where the reactive vibrations interfere with oncoming compressional waves, considerable energy is dissipated, and at times one may be so fortunate as to detect points of increased temperature (probable nodes) upon a drill steel which is cutting ground; and it is no uncommon thing to see a drill steel, in service, break at two points (into three pieces) simultaneously, probably from fatigue because of these vibratory stresses. On the other hand, if these vibrations synchronize at the bit it is quite possible that the chipping forces are greatly augmented, and such an explanation may readily answer those puzzling drill tests in which a dull or broken bit exceeds a finely formed bit in drilling speed. For a long time at Franklin a tally was kept of the different individual drill steels which entered into the testing, with the hope of determining that some particular piece of steel produced the greatest cutting speed, but no conclusion could be drawn from the records, except that the changes in length due to resharpening probably masked any possibility of determining the suitable lengths for maximum efficiency. And it seems quite plausible that such a result should be expected if the possible wave lengths of the compressional vibrations in the drill steels are considered. Probably these reactive vibrations occur to a great extent, as well, in the process of drilling, where the steel dances in the chuck and against the rock, for steel breakage appears equally as high, if not higher, with such a type of machine as with the pneumatic feed, and tests comparing these two types for such effects might prove very interesting as well as instructive.

But still other factors influence the force delivered at the rock. If the anvil block or tappet is not in contact with the drill when the piston strikes, a considerable energy loss occurs through the transference of momentum to several pieces. If the steel is bound in the chuck bushing, a great amount of the energy is absorbed by the friction. If the steel is not straight, it loses energy because of the flexure. If the chuck is badly worn, the axis of the steel does not coincide with that of the drill and there is a loss due to the oblique, eccentric impact. If the steel is tight in the drill hole, or if the friction against the side of the hole is great because of its depth, the velocity of the steel, as a projected body, is lessened and the drilling speed is reduced.

The length of the drill steel is an item generally credited as an important influence, and common opinion supports the idea that the cutting speed falls as the length of the steel increases, although some people, on the contrary, feel sure that the long steels drill the fastest. The tests conducted at Franklin do not lend an unqualified support to either view, for the peculiarities of different types of machines play so important a part. For example, if the air feed is very strong in a stoping drill the additional counteracting weight of a long and heavy steel may so improve the working conditions as to indicate a superiority for the long steel, and if

the air feed is weak the reverse may be true; if the drill steel cuts by virtue of a dancing or "jumper" action, the mass added with length may so reduce its velocity against the rock as to bring it below the amount required for efficient chipping; if the piston normally delivers too heavy a blow for the rock, the drilling speed may be improved by the added inertia of the long steel; and if the steel is always against the rock when a blow is delivered, it is doubtful whether the length of the steel plays an important part unless the permitted decrease in the gauge of the drill bits aids the cutting speeds. It is, of course, to be understood that the above considerations of drill-steel lengths refer to the performances with bit gauges of the same diameter.

The use of an anvil block is considered by some drill designers to necessitate a loss of from 20 to 30 per cent. of the power of a drill, but actual tests do not always indicate such a condition when the identical steel is tested in the same drill with and without a tappet. The results probably depend upon how frequently the tappet is struck when away from the shank of the steel, and also upon the suitability of the machine to the rock, for if its blows are too heavy the intervention of a loose tappet might reduce their force, with a benefit in drilling speed. The use of water at the bottom of the hole ordinarily consumes about 10 per cent. of the cutting speed if there is no tendency for the drill bits to lose their temper, and compressed air for cleaning the holes encourages a greater drilling speed, providing the cushion of water in the bottom of the hole does not have a benign influence in reducing too powerful a blow upon the rock.

The manner in which a drill is rotated has a bearing upon the amount of work accomplished, and with hand-rotated tools, a vigorous rotation with a rapid and wide arc of swing produces the best results; with power-rotated drills it is possible to reach such a speed as to abrade and dull the drill bits, and consequently lessen the drilling speed. It seemed that, with a positive and constant rotation, the axial planes of the cutting edges of the drill bits should be at the same angle with the cut surface as the resultant velocity vector, as estimated for the rotative and striking velocities; and such a bit was tried at Franklin without showing a change in cutting speed, probably because with either bit the chips came out in flakes, as previously described.

In view of the fact that the subject of hammer drills is more or less in its infancy and literature in regard to them is rather limited, it seems desirable to correct at the earliest opportunity any typographical or other errors which, if accepted without investigation, might work to the detriment of the art of drilling. In this connection it seems that some statements should be corrected in the 1910 edition of Eustace M. Weston's book, *Rock Drills,* in the chapter Philosophy of Process of

Drilling Rock, under the sub-heading of hammer drills. In considering the kinetic energy of a blow he states, on p. 139:

"In other words, to double the energy of a blow it would be necessary to double the mass, or weight, if the velocity is the same; *but to double the energy, keeping the mass the same, the velocity must be increased four times.* The weight of the piston hammer of the largest type of drill is 15 lb. The weight of piston, steel, etc., of a piston drill varies from 60 to 125 lb., so that a blow of equal force can be delivered by a hammer drill only by increasing the velocity of the hammer very greatly. This is acknowledged, for as one hammer-drill maker states, the weight of the piston is one-fourth that of a piston drill; but the velocity is four times as great. *To give a blow equal in power it should be sixteen times as great.*"

A mathematical error appears to have been made in the premises of Mr. Weston's argument and his consequent deductions as to the practical impossibility of hammer drills being able to compete with piston drills are quite logical, but probably at fault.

If the kinetic energy of a body, such as a drill piston, is designated by K, its velocity by V, and its mass by M, then,

$$K = \tfrac{1}{2}MV^2 \text{ and } K_1 = \tfrac{1}{2}M_1V_1^2$$

Now if M_1 equals M and K_1 is, say, twice the value K, then,

$$V_1^2 = 2V^2 \text{ and } V_1 = V\sqrt{2}$$

therefore,

$$V_1 = 1.414V$$

So the velocity of the piston in a hammer drill need be only 1.4 times as great as that when the kinetic energy of the piston is cut in half.

Again, in the example comparing the piston weights of piston and hammer drills, Mr. Weston appears in error in stating that the velocity should be 16 times as great, for if the piston of a hammer drill is one-fourth the weight of that of the piston drill the velocity of the hammer-drill piston need be only twice as great as that of the piston drill in order to deliver blows of the same energy; and the hammer drill will also surpass the piston drill since it will strike twice as many of such blows per minute. The necessity of using high air pressures in hammer drills is only incident to the peculiarity of certain drill designs and is not dependent upon the divorcing of the piston from the steel. If we are to consider the shock upon the parts of two drills of equal capacity it is evident that with the shorter piston strokes in hammer drills, with the increased number of blows, whose final striking velocity is equal to that of a piston drill under comparison, the weight of the hammer-drill piston may be less and the energy in each individual blow may be less in order that the same amount of energy per minute be developed. Therefore, the shocks upon hammer-drill parts are more frequent but not as heavy as the shocks upon piston drills of equal capacity.

It is extremely difficult to get adequate figures as to the maintenance of drills unless some special forms are kept, which become to all intents a ledger account of each individual drill, for questions naturally arise as to the cost per foot of hole drilled, the length of time the machine has been in service and has been running, the number of holes drilled, the kind of rock encountered, and the supply of steel used, as well as the drill parts replaced. The New Jersey Zinc Co. uses a system of punched slips for shift bosses' reports and "drill record" slips are a part of the scheme. These are punched in duplicate by the shift boss and filed at the mine office and main office, where the information is transferred to large sheets, of which each one accommodates the record of one machine for a month, and the footings are carried forward so as to indicate the total work accomplished and maintenance of any machine "to date." The repair parts are designated as to whether they are new or old (second-hand) ones, and

AB 1507-500 B'ks 11-14

DRILL RECORD

LEV.	50	100	150	200	250	300	350	400	450	500	550	600	650	700	750	800	850	900	950	1000	1050	1100	1150	M.D.	LONG.	RAISE	D'FT.	GN'S	FELD	CTS.	CLNG
STOPE	859	908	968	1020	1070	1141	1190	1239	1392	1440	1493	1550	1597	1644	1697	1747	1797	1855	1905	1963	2017	2071	2125	2226	2333	2405	O.C.	LIME	GARN	BH'G	BKN
	1000	900	800	680	622	564	500	436	360	290	229	176	123	80	4	78	139	291	348	401	451	535	612	669	716	763	812	ORE	PEG	STP'G	IDLE

JAN.	SEP.	0	DR'L'G		HOLES		FEET		HRS REP. LAB			DRILL STEEL USED						MAKE:-		PIL'R
FEB.	OCT.	1	HRS	MIN	0	0	½	0	*	R	H	1st	2nd	3rd	4th	5th	6th	TYPE:- NO.		
MAR.	NOV.	2	1	10	1	1	1	1	1	1	1	1	1	1	1	1	1	REC'D:- DISCH'D:-		F.W.
APR.	DEC.	3	2	20	2	2	2	2	2	2	2	2	2	2	2	2	2	SENT MACH. SHOP:-		H.W.
MAY		4	3	30	3	3	3	3	3	3	3	3	3	3	3	3	3	REP. PARTS:-	N	O
JUNE	1	5	4	40	4	4	4	4	4	4	4	4	4	4	4	4	4		N	O
JULY	2	6	5	50	5	5	5	5	5	5	5	5	5	5	5	5	5		N	O
AUG.	3	7	6	60	6	6	6	6	6	6	6	6	6	6	6	6	6	REMARKS:-		N
1914		8	7	P	7	7	7	7	7	7	7	7	7	7	7	7	7			S
1915	DAY	9	8	HD	8	8	8	8	8	8	8	8	8	8	8	8	8			E
1916	NGT	10	9	BH	9	9	9	9	9	9	9	9	9	9	9	9	9			W

FIG. 3.—DRILL-RECORD SLIP.

original and subsequent drilling-test records are noted on the same summary sheet.

Figs. 3 and 4 show the record slip and sheet. The locations of drills are by stope-slice co-ordinates, the class of work (whether raising, drifting, stoping, block-holing, or drilling chutes) is indicated, the kind of rock (ore, limestone, gneiss, pegmatite, garnet, or feldspar) is punched, and if the machine is idle, broken, or being cleaned those points are recorded.

The compressed-air rock drill made revolutionary changes in mining methods and in the reduction of mining costs in units of labor per ton of ore, and at Franklin even more marked savings have been made through the development of hammer drills. There, in the days of hand drilling, a total of 8 ft. in three drill holes with varying diameters of 1¾ to 1¼ in. was considered a fair 10-hr. shift's work, and possibly 8 tons of ore would be broken per drill-shift or 4 tons per man-shift. With 3-in. reciprocating rock drills from 20 to 40 ft. of drill holes, ranging in diameter

from 2½ to 1½ in., would be the average work for a 10-hr. shift, although on rare occasions some men might drill as much as 80 or 90 ft. of holes in a shift, and possibly 20 tons of ore would be broken per shift, or 10 tons per man-shift, since two men were needed on a drill. It seems

A B 1625 (C-4)

THE NEW JERSEY ZINC COMPANY

FRANKLIN, N. J.

DRILL RECORD

MAKE; ______ TYPE, ______ SHOP No. ______

PURCHASED ON REQ, A B ______, 191 IN SERVICE; ______ 191 .

CYLINDER DIAM.; ______ inches PISTON STROKE ______ inches PISTON WEIGHT, ______ lbs ______ oz.

AVGE. DRILLING SPEED to date ______ inches per min. AVERAGE AIR CONSUMPTION; ______ cu ft. per min. free FACTOR; ______

ORIGINAL — DRILLING SPEED ______ inches per min. AIR CONSUMPTION ______ cu ft per min. free FACTOR; ______

PRESENT ( ) — DRILLING SPEED ______ inches per min AIR CONSUMPTION ______ cu. ft. per min. free FACTOR, ______

DATE		WORKING PLACE		DRILLING TIME		NO HOLES DRILLED	FOOTAGE DRILLED	DRILLING SPEED IN FT PER HR	MATERIAL DRILLED	PCS DRILL STEEL USED						HRS REPAIR LABOR			REPAIR PARTS (Maker's Symbols)		M'T'CE CHARGES		
		Lev	Loc	Hrs.	Min					1st	2nd	3rd	4th	5th	6th	Special	$2.25	$1.85			SUPPLIES	MINE LABOR	MACH. SHOP LABOR
Brought Over																							
1st	DAY NGT																			N O			
2nd	DAY NGT																			N O			
3rd	DAY NGT																			N O			
4th	DAY NGT																			N O			
5th	DAY NGT																			N O			
6th	DAY NGT																			N O			
7th	DAY NGT																			N O			
8th	DAY NGT																			N O			
9th	DAY NGT																			N O			
10th	DAY NGT																			N O			
11th	DAY NGT																			N O			
12th	DAY NGT																			N O			
13th	DAY NGT																			N O			
14th	DAY NGT																			N O			
15th	DAY NGT																			N O			
16th	DAY NGT																			N O			
17th	DAY NGT																			N O			
18th	DAY NGT																			N O			
19th	DAY NGT																			N O			
20th	DAY NGT																			N O			
21st	DAY NGT																			N O			
22nd	DAY NGT																			N O			
23rd	DAY NGT																			N O			
24th	DAY NGT																			N O			
25th	DAY NGT																			N O			
26th	DAY NGT																			N O			
27th	DAY NGT																			N O			
28th	DAY NGT																			N O			
29th	DAY NGT																			N O			
30th	DAY NGT																			N O			
31st	DAY NGT																			N O			
TOTAL FOR MONTH																							
Carried Forward																							

FIG. 4.—SHEET FOR RECORDING DRILL PERFORMANCE AND COSTS.

that as a rule a greater tonnage per foot of hole was obtained with hand drilling because of the fact that, rather than dismount and reset heavy drill columns, machine men would tend to place as many holes as possible from one set-up, therefore many holes were placed disadvantageously

for breaking efficiency. Another cause, which would contribute to the same results, would be the difficulty of starting holes with piston drills on uneven sloping faces, so that holes were frequently deflected from the direction in which they were supposed to be placed. These figures would lead to the rough estimate that 2½ times as much tonnage per drilling man-shift was accomplished by piston drills as by hand drilling.

With hammer drills 80 to 100 ft. of 1¾ to 1¼ in. drill holes are placed by one man in a 10-hr. shift, about 150 to 200 tons of ore will be broken per drill-shift and the same amount per drilling man-shift, or 15 to 20 times the amount broken per man with reciprocating drills. Of course the entire credit for such increase in tonnage cannot be given to the type of drill, for improved organization, system of working, and supervision have undoubtedly played an important part; but the greater mobility and flexibility of the light hammer drills have permitted and encouraged a more efficient placing of drill holes; have cut in half the labor necessary to run a drill; and permitted a more effective supervision and mining scheme. The actual tonnage broken per man working in a stope will not be so high, comparatively, since it has been found worth while to place additional men in stopes to sledge and block-hole large chunks of ore, which were formerly often allowed to become buried and so proved obstacles to high tramming efficiency by blocking chutes through which the shrinkage stopes were drawn into tram cars.

Table II shows the gains which have been made with the adoption of hammer drills by the New Jersey Zinc Co. It is to be regretted that no records of tonnage and labor were available for earlier years, so as to cover the average efficiencies before the advent of hammer drills and back to the days of hand drilling. The different divisions of mining work are classified in this table as drifting, raising, stoping, and open cut or quarry, and it may be interesting to summarize the important features, reducing the labor to an hourly basis, inasmuch as a change was made from a 10-hr. to an 8-hr. shift basis in July, 1913.

Drifting.—There have been no radical changes in the placing of the drill holes in drifts since the adoption of the air-feed hammer drill for this work, but one man with a single machine is now placed in a heading; he is instructed to "pull" a "round" each 8-hr. shift, stopping overtime if necessary, and to accomplish an advance of 3½ to 4 ft. per round. Two men operating a reciprocating rock drill formerly made an advance of a 5 to 6 ft. round in five 10-hr. shifts. So the drilling labor (runners and helpers) per foot of advance averages 18.4 hr. for the entire mine during the year 1910, when reciprocating rock drills were solely in use. As shown by the average for 1913, hammer drills have reduced this figure to 5.3 hr. per foot of advance, or about one-third the former labor of drilling and blasting. The explosive costs have also been reduced by

the use of hammer drills from the figure of $1.84 per foot of drift during 1910 to $1.40 per foot in 1913, for two probable reasons.

First, hammer drills permit the placing of drill holes smaller in diameter than those bored by reciprocating drills, so that an unnecessary amount of explosive is not required merely to fill the holes sufficiently to distribute the force of the explosion.

Second, the flexibility and ease of rigging the light hammer drills permit and encourage a more efficient placing of drill holes. The almost exclusive use of 1 by 8 in. explosive cartridges now, as contrasted with the 1⅛ by 8 in. cartridges formerly used, demonstrates the first contention, for in terms of 1-in. powder, the equivalent of 36.8 sticks per foot of drift was used in 1910, and 34.6 sticks per foot in 1913. The drill shifts per foot of advance have been lowered from 0.74 in 1910 to 0.33 in 1913, and the corresponding drill hours from 7.4. to 3.0.

The different drifts may vary in size from 6 by 7 ft. to 8 by 11 ft. in section, and perhaps 7 by 8 ft. is an average section. Because of the compact, tough nature of the ground, it requires from 20 to 30 drill holes in a round, and 24 would be a fair average, so the drilling operation is an important factor of the drifting costs. The following comparison of the average drifting costs for each year shows the saving which has been possible because of hammer drills; but only the cost of drilling labor and explosives is considered. The record drift in 1913 was driven for $2.06 per foot.

	1910	1911	1912	1913
Drifting cost per foot..............	$5.33	$4.92	$3.35	$2.70

Raising.—In 1908, using 2¼-in. piston reciprocating rock drills, 0.7 ft. of 6 by 6 ft. raise per 10-hr. drill shift was made with a labor expense of 28.5 man-hours per foot of raise. About 27 ft. of drill holes were placed per shift, 24 holes were placed in a round, and 16 lb. of explosives were used per foot of raise advance, at a cost of $2.70 per foot for supplies. Since labor was then paid $2 and $1.55 per 10-hr. shift, the total cost of raising was approximately $7.50 per foot of advance.

During the same year, 1908, hammer drills were introduced, and an advance of about 1.5 ft. per drill-shift was made with a labor expense of 13.3 man-hours per foot of advance. About 50 ft. of drill holes were placed per shift, 24 holes per 5-ft. round, and 10 lb. of explosives were used per foot of advance, at a cost of $1.75 per foot for supplies and a total cost of $4.10 per foot of raise, or only 55 per cent. of the cost with the reciprocating rock drills.

The development of hammer drills with increased drilling speed permitted the reduction of the drilling labor to 7.8 hr. per foot of raise advance, and the explosives cost to $1.54 per foot of raising done in the year 1910; and a further reduction to 4.8 hr. of drilling labor during 1912,

TABLE II.—*Annual Comparisons of Mining Efficiencies with Piston Drills and Hammer Drills*

Date	Drifting										
Year	Footage Advance	Drill Shifts per Foot Advance	Type of Drills, Per Cent.		Per Cent. Size of Powder, 50 Per Cent. Strength		No. Caps per Foot	Powder, in Sticks per		Explos. Cost per Ft. Advance	Runners and Helpers per Foot Advance
			H. D.	P.	1⅛ in.	1 in.		Foot Advance	Drill Shift		
1908[a]	494	1.45	.....	.....	.....	.....	.....	36.4	25.2	.....	.....
1909	4,890	1.31	.....	.....	.....	.....	.....	32.7	25.0	.....	.....
1910	4,909	0.74	.....	99.2	73	27	5.43	31.1	42.5	$1.84	1.84
1911	2,814	0.63	.....	99.2	96	4	4.95	26.5	41.9	$1.59	1.71
1912	374	0.44	100	....	11	89	7.40	36.4	89.6	$1.61	0.84
1913	905	0.33	100	....	.....	100	5.56	34.6	106.0	$1.40	0.59

[a] Last 4 months.

Date	Raising										
Year	Footage Advance	Drill Shifts per Foot Advance	Type of Drills, Per Cent.		Per Cent. Size of Powder, 50 Per Cent. Strength		No. Caps per Foot	Powder in Sticks per		Explos. Cost per Ft. Advance	Runners and Helpers per Foot Advance
			H. D.	P.	1⅛ in.	1 in.		Foot Advance	Drill Shift		
1908	715	1.18	.....	...	.....	.....	.....	27.7	23.4	.....	.....
1909	5,446	0.90	.....	...	.....	.....	.....	29.3	32.4	.....	.....
1910	4,257	0.44	97.0	...	16	84	5.11	27.4	62.7	$1.54	0.78
1911	1,865	0.23	98.5	...	8	92	4.38	21.5	92.5	$1.04	0.65
1912	1,311	0.17	100.0	...	2	98	3.46	20.1	115.0	$0.93	0.48
1913	2,306	0.20	100.0	...	1	99	3.94	21.4	106.2	$1.03	0.58

Date	Stoping (active)												
Year	Net Tons Ore Broken	Type of Drills, Per Cent.			Net Tons Broken, Excl. B. H. Drills per Drill Shift	Per Cent. Size of Powder, 50 Per Cent. Strength		No. Caps per Ton	Powder in Sticks per		Explos. Cost per Net Ton Broken	Net Tons Broken per Man in Stope	Remarks
		B. H.	H. D.	P.		1⅛ in.	1 in.		Net Ton	Drill Excl. B. H. Shift			
1908	57,000[b]	....	....	....	23.4	....	....	.....	0.76	17.7 19.3	......	....	
1909	361,000	7.0	24.5	68.5	20.5	....	....	.....	1.01	20.7	......	....	
1910	337,000	12.7	62.3	25.0	38.1	16	84	0.310	1.025	39.0	$0.055	13.2	
1911	386,000	30.1	55.8	14.1	121.0	11	89	0.356	0 77	95.8	$0.041	20.3	
1912	427,000	35.5	62.0	2.5	195.0	6	94	0.415	0.86	108.3	$0.045	25.6	
1913	494,330	30.3	69.7	....	170.0	2	98	0.420	0.89	233.0	$0.049	26.7	

[b] Two months, estimated.

Date		Filling											
Year	Net Tons Rock Broken	Type of Drills, Per Cent.			Net Tons Broken, Excl. B. H. Drills per Drill Shift	Per Cent. Size of Powder, 50 Per Cent. Strength		No. Caps per Ton	Powder, in Sticks per		Explos. Cost per Net Ton Broken	Net Tons Broken per Man	Remarks
		B. H.	H. D.	P.		1⅛ in.	1 in.		Net Ton	Drill Excl. B. H. Shift			
1910	25,400	58.6	14.2	27.2	114.2	28	72	0.922	0.745	(35.2)[c] 85.3	$0.051	24.4[c] (4.9)	
1911	64,500	79.2	1.6	19.2	355.0	29	71	0.787	0.614	(45.3) 278.0	$0.042	62.0 (50.6)	
1912	107,800	62.0	11.4	26.6	228.3	29	71	0.571	0.719	(62.4) 164.5	$0.040	54.2 (46.0)	
1913	250,490	45.7	54.3	....	236.0	39	61	0.524	0.712	(91.3) 168.0	$0.042	82.2 (51.0)	

[c] Bracket figures include block-holing shifts, and labor of stopers, runners, block-holers and helpers; others rated against runners only.

and an explosive cost of $0.93 per foot, although the wages were $2.20 and $1.70 per 10-hr. shift. These costs rose slightly in 1913, since wages rose to $2.25 and $1.85 for 10-hr. shifts, and in July of the same year the working hours were lessened from 10 to 8 and the hourly wage was increased to $0.281 and $0.231. However, the cost per foot was then only 5.2 hr. of drilling labor and $1.03 per foot for explosives. About 18 drill holes are now placed to pull a 5-ft. round and two men are expected to blast a round each 8-hr. shift and are each paid 11 hours' time for performing the task.

The average raising costs for operating labor and explosives have been as follows:

	1910	1911	1912	1913
Raising cost per foot............	$2.95	$2.31	$1.88	$2.22

The record short raise (of about 50 ft. in length) for 1913 had a cost of $1.65 per foot, and the record long raise (about 100 ft. long) had a cost of $2.09 per foot, with explosive costs, respectively, of $0.77 and $0.99 per foot of raise.

Stoping.—In 1909, when about 74 per cent. of those drills placing holes in the solid orebody were of the reciprocating type of 3-in. piston diameter, the ore production averaged about 20 net tons of ore broken from the solid per 10-hr. drill shift, with an equivalent of 1.1 sticks of 1 by 8 in. of 50 per cent. dynamite per ton of ore.

In 1910, when about 72 per cent. of the producing drills were air-feed stoping (hammer) drills, the tonnage per drill shift rose to 38 net tons with about the same amount of explosive (which cost $0.055 per net ton of ore broken), and 13.2 tons were broken per 10-hr. shift of men

working in stopes, or 1.32 tons per hour. Although there is no record of the breaking labor prior to this year, the fact remains that in the actual running of the drills only one man was used with a hammer drill while two men were employed with each reciprocating drill.

In 1911, when the hammer drills were about 80 per cent. of the total, the stoping efficiency profited by the improvements in the drilling speed of the hammer drills, and 121 net tons were broken from the solid per drill-shift with about 0.8 stick of 50 per cent. 1 by 8 in. dynamite per ton (at an explosive cost of $0.041 per ton), and 2.03 net tons were broken per man-hour of men working in stopes.

In 1912, when about 98 per cent. of the stoping drills were hammer drills, 195 net tons were broken per drill-shift with about 0.87 stick of 50 per cent. 1 by 8 in. dynamite per ton (at an explosive cost of $0.045 per ton), and at the rate of 2.56 net tons per man-hour in the stopes.

In 1913, when all the stoping drills were hammer drills, the length of the working shift was reduced from 10 to 8 hr. in the middle of the year and the tonnage broken per drill-shift fell proportionately to 170 net tons, but remained at approximately the same hourly rating as for the year 1912. However, the tonnage broken per man-shift in the stopes increased slightly to 26.7 net tons (at an explosive cost of $0.049 per ton), with the consumption of 0.89 stick of 50 per cent. 1 by 8 in. dynamite per ton. The tonnage broken per man-hour was 2.97, which showed a steady gain over previous years.

It should be noted that the explosives charged against stoping include those used by the trammers in blasting ore in the chutes, and thus represent all the dynamite necessary to reduce the ore to the proper size for being handled through chutes and in the mill.

Opencut.—In order to provide broken rock for filling material to fill empty stopes to support the remaining orebody, "mill-holes" are developed in limestone country rock at the surface. For some years it was the practice to use 30-ft. bench-holes in the opencut for quarrying the rock, both 3-in. and 3½-in. reciprocating rock drills being used. It took steady work for two men to sink one 30-ft. hole in a 10-hr. shift, and their work was hazardous because of the inconvenient localities where set-ups were made, and because of the clumsy weight of their machines and the long, heavy drill steels which were handled. After the success of hammer drills in the underground mining operations, they were tried in the opencut work in 1912. Small holes were drilled to an average depth of 16 ft., and were given lighter burdens than had previously been the practice, for the object was to distribute the dynamite more evenly in the rock, as contrasted to churn-drill or mammoth blasts. In the tough, crystalline Franklin limestone this application of hammer drills to quarrying has proved superior to the heavy or mammoth blasts, for the same tonnage can be produced from a bench

with a saving of labor and powder, since a great amount of expensive block-holing is avoided. A machine will drill about 100 ft. of holes in a shift with a heavy hammer drill and two men can drill only 30 ft. with a rock drill.

Discussion

T. E. Sturtevant,* New York, N. Y.—Mr. Tillson's paper on hammer drills is of much interest to me. Early in 1909 I was invited to make tests of our drills at the Franklin mine of the New Jersey Zinc Co.

The first drill that we tested gave very poor results. This was a drill having a hammer piston 1¾ in. in diameter and 3¼-in. stroke, the hammer piston weighing about 6 lb., the 1¾ in. diameter being exposed to pressure at either end. Although this drill has given excellent results as a sinking drill, its performance as a stoping drill was poor. This was one of the surprises of the limestone as spoken of by Mr. Tillson.

This led me to further investigation, and a second drill was developed, with a much lighter piston and a somewhat longer stroke, that gave better results. In the design of this drill, no attention was paid to the cost of construction, our whole attention being given to the increase in drilling speed and reduction of the quantity of air used. The first drill of this type had a piston weighing about 2½ lb. and a stroke of 3¼ in. The piston was made with a hammer-bar extension, and the cylinder was made with two diameters of the bore, one in which the hammer-bar extension moved and the larger one in which the piston head moved. The air was taken in at either end of the larger bore, thus giving a greater area to the piston on the forward stroke than on the rearward stroke. The pressure area of the piston for the forward stroke was the 2 in. diameter. The pressure area for the inward stroke was 2 in. diameter less the 1⅜ in. diameter of the hammer bar. The performance of this drill was much better than that of the first drill, the air consumption being less and the drilling speed slightly greater. We then increased the length of the stroke to 3½ in., this giving better results. Again we increased the stroke to 5½ in. without any gain over the drill with the 3½-in. stroke. Through a number of tests we gradually reduced the length of the stroke until we found that the highest drilling speed was attained with a stroke of 3⅞ in.

While I note the factor of efficiency as given by Mr. Tillson has been exceeded by other makes of drills, the work of our drills in the mine was satisfactory. The usual day's run was from 14 to 18 6-ft. holes per shift, and in two separate instances there were 26 6-ft. holes drilled and the heading fired in one shift.

The question of the loss of efficiency with an anvil block, as well as that of the position of the steel in relation to the rock during the moment of

* Chief Engineer, McKiernan-Terry Drill Co.

impact, is fully covered by Mr. Tillson. In our drills, I can note no difference in the drilling speed with or without the anvil block. We find that with the steel loose enough so that it will ring at each impact of the hammer, the highest drilling speed is attained.

The trial of drills by the mine management of the New Jersey Zinc Co. has always been conducted with the utmost fairness, and at no time could these trials of drills be considered a contest, as strict privacy was given to each manufacturer, and while careful record was kept of the performance, these data were not available to any, except the owners of the drills and the mine management. I have been a frequent visitor at the mines since 1909, and I have at no time there seen the construction of a competing drill or from them had any information as to its performance. This, I believe, is their general rule to all who have drills at their mine for trial.

The drill manufacturers can furnish hammer drills with a high cutting speed, but the users have the human element to contend with and I believe that the best results can be attained with a bonus system.

H. M. Crankshaw, Lansford, Pa.—Mr. Tillson stated that the efficiency of these drills has been improved about four times. Does all of this improvement come from the drills, or has any of the improvement come from the air? It has been my experience that the capacity of a drill can very often be doubled by increasing the capacity of the compressors and by putting in good air lines. They have the same plant now that they had in 1907, but the question arises, are the air lines any better now than they were in 1907? These figures, I take it, are figures over a series of time, and not figures taken from a test.

There is another rather curious point here, about the bit with the broken wing. I have seen it stated that a common X-bit with the two edges which were in line raised 1/8 in. higher than the other edge, would give better results in drilling than the ordinary X-bit. Did Mr. Tillson follow up this seeming advantage he got with the broken wing?

Benjamin F. Tillson.—The question of the increase in drilling efficiency is covered by this summary of the tests. You can follow it in the tests. The air pressure at the drill during the test is tabulated, and approximately the same air was used in the tests, so that the question of drill-plant equipment has not in any way affected it. As a matter of fact, all of these tests have been conducted with the same installation of air line in our shafts and the same pressure in the mine.

In regard to the broken bit, I probably missed making the point I intended to. It was that even though the bit was broken or shattered, or soft or plastic, yet some particular drill steels might drill faster than other drill steels with perfect bits and under the same standard testing

conditions of machine, air pressure, ground, and drill gauge. Although many factors enter into the problem it seems that the synchronizing of the vibrations in the drill steel may be such as to increase the drilling speed, even though the bit were broken or shattered, to a figure in excess of the customary speeds.

R. M. Catlin, Franklin Furnace, N. J.—At first sight, these tests of Mr. Tillson's seem to indicate a path along which some valuable results might be arrived at, but as these tests progressed further, the number of variables and unknown quantities in the equation became so great that it rather leaves the question somewhat as it was in the beginning.

At first, it seemed clear that if the drill point was pressing against the rock at the instant the blow was delivered, or, more correctly, at the instant that the impulse liberated by the blow reached that point, the greatest cutting effect would occur; that is to say, that in a drill it takes an appreciable time for the impulse to travel through the length of the metal. Now, if we have the hammer striking the drill when that impulse is rising, and the drill is pressing against the rock, we get the maximum cutting effect, whereas if the time is so arranged that the blow of the hammer meets the impulse coming back, you get the work done at the other end of the drill. In actual practice we have seen that as you put your hand on the drill you find a point which is intensely hot, and a few inches farther down quite a moderate temperature, and still farther down another center of temperature, which evidences the points at which, I think you might say, these two impulses overlap each other, nodes, so to speak. It seems that if one could test the steel as to its capacity for transmitting vibrations, and from a pile of steel select certain drills that had the same capacity, and then regulate his machine so as to deliver the blow at the right time, that with a very poor drill you could get a very remarkable result.

That seems to be so. The next thing is to work it out. I know of no way of determining the capacity of steel for transmitting impulse except to actually test each individual piece, and then, after you have run a little while, that capacity changes. If you are running it at a velocity of 1,800 blows a minute, a few blows more or less will change the rythm of the whole thing, so that the point is more of academic interest than of special value. I merely mention it to account for some of the vagaries which Mr. Tillson mentioned. It obviously cannot be better to drill with a dull drill than a sharp one, but the cause is more obscure than it seems, I think.

L. C. Bayles,* Phillipsburg, N. J. (communication to the Secretary†).—This paper brought up several points of interest to any one connected

* Ingersoll-Rand Co.

† Received Mar. 23, 1915.

with mining, but the most important is the determination of the relative desirability of a drill, as practiced by the New Jersey Zinc Co. for the past six years.

On pp. 246 and 247, Mr. Tillson develops what appears to be a perfectly fair formula for making such comparisons. But as it is impossible to determine the upkeep by a short test and since both this and the first cost give such a small figure when divided over the total footage of these machines, Mr. Tillson is no doubt justified in omitting "M" and reducing the formula to the simple expression "Factor of desirability"

$$F = \frac{1}{D+P}$$

With this simple and fairly accurate formula available, it is hard to understand why the New Jersey Zinc Co. should have made up and adopted the empirical formula $F = \frac{1}{DP}$ which is no simpler to use and is obviously wrong, as it gives too much importance to the relatively small item of the value of the compressed air used.

On p. 249, Mr. Tillson takes two hypothetical cases and shows that their relative comparison remains about the same when compared by one or the other formula. But unfortunately he assumed two cases which both showed the same mechanical efficiency, that is to say, drilling an equal amount per unit of power input. Under such conditions naturally the fastest drill would show the highest "Factor of desirability" no matter which formula was used.

If Mr. Tillson had assumed that the slow drill would do 8 in. per minute, but that the other figures all remained the same, he would have seen the difference in the answers obtained by the two formulas.

Using $F = \frac{1}{D+P}$ we would find the fast drill was 17 per cent. more desirable. But using the empirical formula it would look as though the slow drill was 7 per cent. the best.

The above assumptions are entirely within the limits of possibilities. If we select two drills from the table on p. 245, say the ones marked "M" and "XO," we get entirely different results according to the method of calculation. Using $F = \frac{1}{D+P}$ we get "XO" or the fast machine about 4 per cent. more desirable. But by the empirical formula it appears as if the slow machine marked "M" was close to 20 per cent. the best.

Mining Methods of the Arizona Copper Co.

BY PETER B. SCOTLAND, MORENCI, ARIZ.

(New York Meeting, February, 1915)

THE mines of the Arizona Copper Co. are situated in the Morenci-Metcalf copper district in southeastern Arizona. This copper-bearing district covers a triangular mountainous area of about 3 square miles, rising abruptly from the gravel plateau of the surrounding country. In this area, an immense intrusion of porphyry has displaced, shattered, and engulfed the shales, limestones, and quartzites formerly bedded on the basal granite.

Outcrops of oxide ore were discovered in 1873, and these surface ores were found to connect in depth with large bodies of high-grade oxide ores in the limestones and shales. The mining of these high-grade ores continued until about 1893, when their rapid exhaustion led to the discovery and exploitation of the low-grade sulphide ores in the adjoining porphyry.

Methods of concentration were then devised and the low-grade ores finally rendered profitable. The Arizona Copper Co. and the Detroit Copper Mining Co., operating in the same district, were the pioneers in the successful development of the low-grade "porphyry" properties of to-day.

The largest mines of the Arizona Copper Co. are the Humboldt and Clay at Morenci, and the Coronado, Metcalf, and King at Metcalf. These mines produce about 4,000 tons daily of concentrating ore containing from 2.5 to 2.8 per cent. copper.

The topography of the country is very rugged and mining operations are conducted principally through adit levels. A few of the mines are opened by shafts, the deepest being 1,100 ft., in the Coronado mine.

ORE OCCURRENCE

The low-grade sulphide ore occurs as: (1) disseminated deposits in the main stock of quartz porphyry; (2) lode deposits in fault fissures in granite. The high-grade oxide ores were found as tabular deposits in limestone and shale close to contacts with porphyry dikes. Low-grade oxide ore formed the outcrop of some of the sulphide orebodies; the majority of the outcrops, however, were barren.

Orebodies

Unlike most of the newer low-grade copper mines, the porphyry ore of this district does not occur as one large deposit. The main porphyry stock is broken up by the inclosure of large masses of sedimentary rocks resulting in a number of separate orebodies in dikes. Other orebodies have formed along fault-brecciated areas in porphyry or in granite, and the rock outside of the influence of the fault is barren or only weakly mineralized.

The diversity in size of some of the orebodies is shown by the following dimensions:

	Width Feet	Length Feet
Humboldt first orebody....................	80	600
Humboldt second orebody..................	200	700
West Petaluma orebody....................	6	450
Coronado orebody..........................	35	1,750
King orebody................................	25	650

The depth from surface to which the orebodies have been proved to be of commercial value varies from 450 to 800 ft.

Mining Methods

The method of extracting the ore is dependent upon the nature of the ore occurrence. Each orebody is a problem in itself. All influencing factors, as size, shape, position, nature and grade of an orebody, and character of the inclosing walls, are fully considered before the method of extraction is decided. The ore reserves, though large, are not so immense that mining methods involving a sacrifice of ore or its dilution by waste for the sake of cheaper mining costs have ever been used. The following examples are typical of the mining methods in use.

Opencast

The oxide ores found close to the surface on Metcalf Hill are extracted by opencast work. The amount of ore available for open work has never been sufficient to justify the large expenditures needed to install mechanical appliances such as steam shovels; hand work has always been employed.

Benches or terraces up to 30 ft. in height are carried into the orebody and the overburden and ore broken alternately. To prevent admixture of ore and cap rock, heavy blasts are seldom used.

The ore or waste is loaded by hand into cars of 25 cu. ft. capacity, or shoveled into chutes brought up from a lower level to the floor of the working place. In this handling of the ore, a preliminary sorting out of

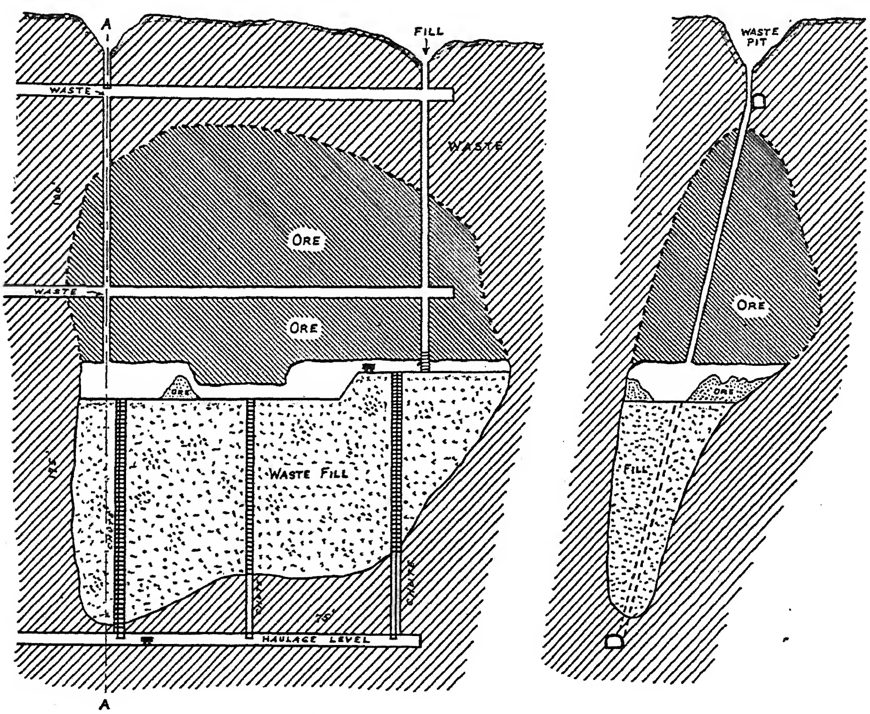
waste rock is made and this is completed on the sorting screens erected over the storage bins.

When conditions permit, mill-holing is used for the more rapid removal of the waste or ore.

The fluctuations in output of ore caused by the alternate handling of overburden and ore are reduced by keeping open a large number of working places.

If the orebody has been proved below the depth of economical open work, disposal of the overburden can be economically made by using it as filling material in stoping the ore on lower levels.

The dry climate and mild winters are favorable to opencast mining in Arizona.



Longitudinal Section. Cross-Section A-A.

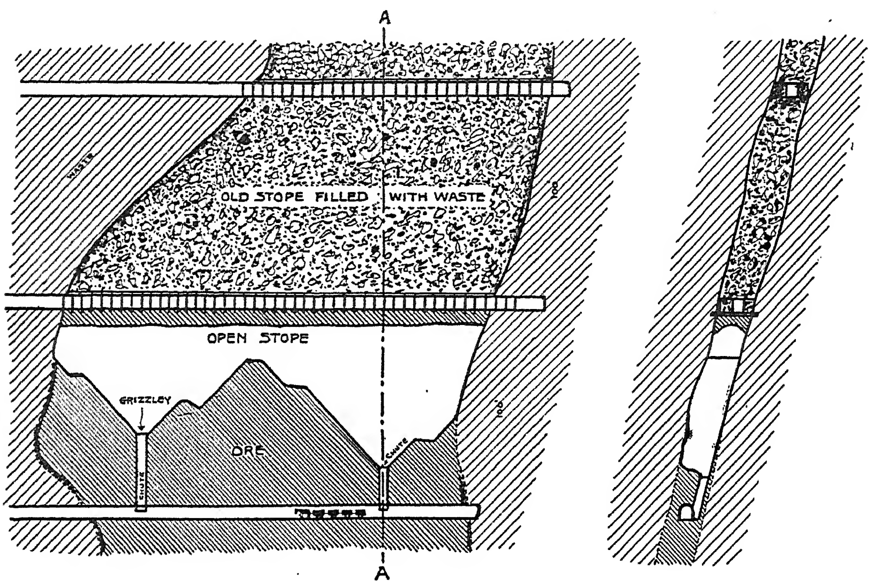
FIG. 1.—OPEN STOPING AND FILLING.

Open Stoping and Filling

Where the ore is hard and stands well, a system of open stoping without timber is employed with subsequent filling. (Fig. 1.)

The bottom of the oreshoot having been determined, it is opened for its full length and width and to a height of 15 to 20 ft. While this is proceeding, one or more raises are driven to the surface or to the level

above for ventilation and waste filling. Chutes and ladderways are then erected from the tramming level and the stope filled with waste to within 8 ft. of the roof. Slices of from 15 to 20 ft. in height are then mined overhand along the whole length of the oreshoot, and waste filling follows behind the working face. The filling is best distributed from the waste chute by means of a light car on a movable track. Mechanical means such as scrapers have not proved successful. Excepting a few log cribs or square sets placed beneath loose portions of the roof, no timber is used.



Longitudinal Section. Cross-Section A-A.

FIG. 2.—UNDERHAND STOPING.

Where the orebody is on or close to an important tramming road, stoping is started above the level of the road and a shell of ore 10 to 15 ft. thick is left to protect the roadway.

Several stopes in Metcalf mine worked by this method were so wide (up to 300 ft.) that pillars representing about 5 per cent. of the orebody were left to support the roof. If waste filling is cheaply obtained, either from surface stripping or from glory holes, the system is very economical, but the danger to the men from the unsupported roof limits this method to only the best standing ground.

Underhand Stoping

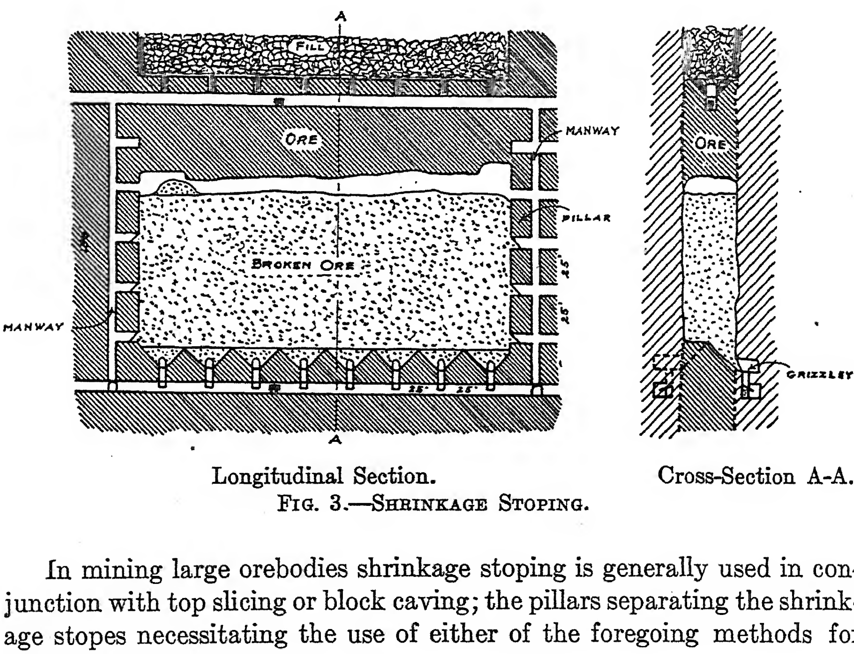
In narrow veins with firm walls, underhand stoping directly into chutes is employed. (Fig. 2.)

Untimbered chutes are carried to the top of the orebody at intervals of 25 to 50 ft. Around the top of these chutes underhand mining commences, the broken ore falling directly into the chute. The working floor is always kept cone shaped to avoid shoveling the broken ore. A grizzly of logs prevents very large pieces of ore from entering the chute.

This method is only practicable under a sound roof, as in the process of stoping the height of the roof is constantly increasing and its observation made more difficult.

Shrinkage Stoping

In this method, the broken ore is allowed to accumulate in the stope to support the walls and form a working floor for mining the roof. Practically no timber is used and the ore is drawn from chutes at the bottom of the stope directly into the mine cars.



Longitudinal Section. Cross-Section A-A.

FIG. 3.—SHRINKAGE STOPING.

In mining large orebodies shrinkage stoping is generally used in conjunction with top slicing or block caving; the pillars separating the shrinkage stopes necessitating the use of either of the foregoing methods for their removal.

The application of this method at the Coronado mine is shown in Fig. 3. The vein is about 35 ft. wide and is stoped in lengths of 150 ft. with pillars 30 ft. long between the stopes. The overhand mining is confined to the ends of the stope with the object of leaving a sag or belly in the roof of the ore. This finally breaks off by its own weight and, if it does not break by its fall, it is "bulldozed" on top. As stoping progresses, about one-third of the broken ore must be drawn out to allow working space below the roof. Should the roof become dangerous to work beneath, the ore can be broken down by underhand stoping from a

sub-level driven 25 to 30 ft. above the stope. Access to this sub-level is obtained from the ladderways in the end pillars.

The breaking of ore is stopped 25 to 30 ft. below the bottom of the overlying stope and the broken ore then rapidly drawn through the chutes. The old waste filling in the overlying stope is tapped and run into the empty stope. When full and leveled off, the shell of ore left in the roof is gained by square-set timbering. The pillars between the stopes are then removed by top slicing.

The chute arrangement at the bottom of the stope varies in different orebodies. At the King mine, the ore breaks fine and can therefore be pulled directly through the chutes. At the Coronado mine, the ore is harder and cannot be run directly into cars without much block-holing in the chutes. Two different methods are used to allow of the ore being broken by hand. In the first system, haulage roads are run in the foot wall parallel with the orebody and inclined chutes put up at 30-ft. intervals into the stope. Over each chute is a grizzly on which the laborer stands to rake down the ore and break the large pieces passing into the chute. Another system employed is to run a level, timbered two sets high, through the center of the orebody. On the sides of the upper set of this drift inclined raises are made to the floor of the stope which is opened 20 ft. above the level. The broken ore passing through these raises is loaded directly into the cars beneath by opening the lagging on top of the first set.

The method of completion of a stope depends upon its depth from surface or the condition of the upper workings. If on top of the orebody, the stope can generally be carried to cap rock and then drawn quickly without difficulty. The danger of an air blast from the sudden caving of an empty stope makes subsequent filling always advisable.

Where the roof of the stope is too weak to leave an unsupported shell, the roof is carefully arched and timbered with long heavy stulls from wall to wall, before the broken ore is drawn.

A method was tried of laying a heavy mat of timbers on top of the broken ore, caving the waste above on top of this mat and drawing the ore from beneath. It was found that if the mat was caused to descend more than 30 ft. it became so broken up that a serious dilution of the ore with waste resulted.

A vital objection to this system developed in the Coronado mine. The ore from this mine contains a larger amount of pyrite than is usual in the district, and when the broken ore was drawn from the stopes it was found to be so highly oxidized that the system had to be abandoned.

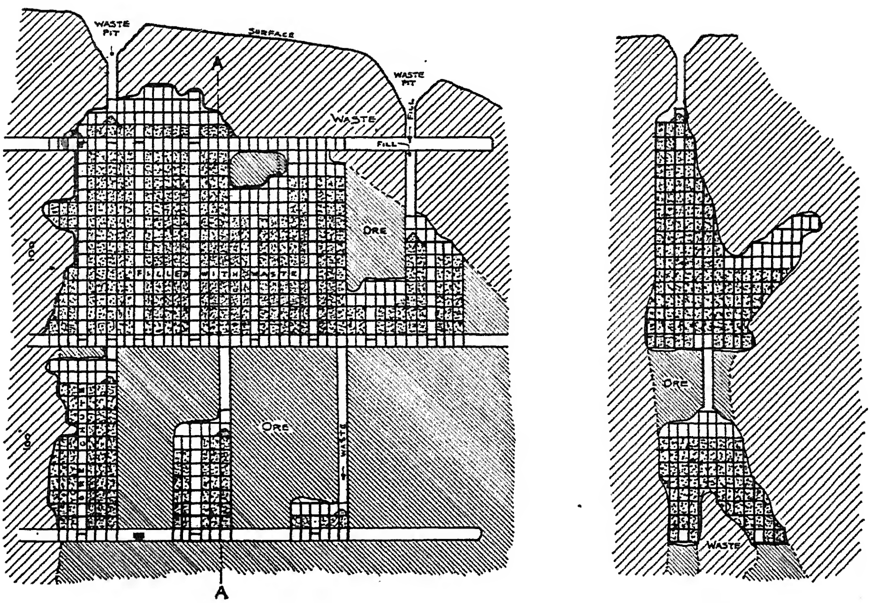
Square Set and Fill

The principal mining method employed for many years in the district was square set and fill. (Fig. 4.)

The expense of using this system in the large orebodies led to its disuse and the substitution of top-slicing methods. An excessive amount of timber had to be used in reinforcing the square sets to prevent serious caves, and the maintenance of raises to surface for waste became more difficult as the workings extended.

The square-set method is, however, still employed in the extraction of orebodies of irregular shape, and in the first stage of top-slicing. Except in minor details, square-set practice here does not differ from that of other districts.

The set of timber comprises an 8 by 8-in. cap, 6 by 8-in brace, and a



Longitudinal Section. Cross-Section A-A.

FIG. 4.—SQUARE-SET METHOD.

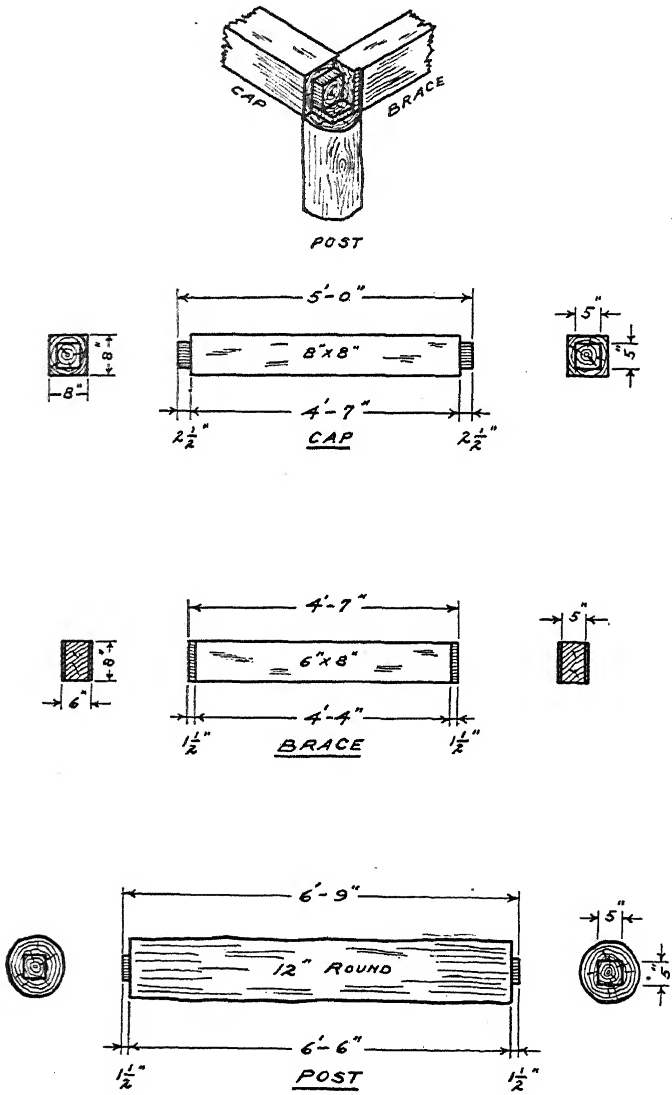
6½-ft. round post. The set allows of the extraction of 178 cu. ft., or 14 tons, of ore. (Fig. 5.)

Waste filling is obtained from raises to surface or to old stopes above and the floors are kept filled to within one set of the roof.

As the high cost of timbering in square-set work is due to the necessity for reinforcing caps and braces that are overloaded, stopes of small floor area are opened and worked rapidly.

A change in the square-set practice has recently been made by using sets 10 ft. high, allowing of the extraction of 250 cu. ft. of ore. The stope posts of 6½ ft. height have been replaced by 9½-ft. posts; the other set members have not been changed.

The increase in height of the set causes only a small reduction in its strength and reduces the cost of timbering about 15 per cent. In a stope, the weight of the roof is carried by the caps, braces, and lagging and through these transmitted to the posts. The crushing strength of



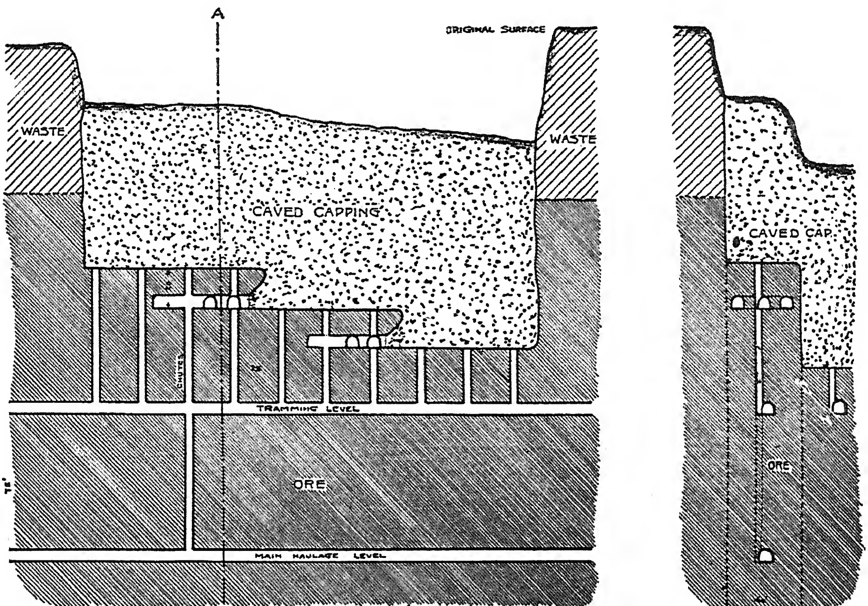
Sets are 5 ft. square and 7 ft. 2 in. high, centers.
FIG. 5.—SQUARE-SET FRAMING.

the posts is so much greater than the transverse strength of the cap and brace that an increase in the height of the post does not materially reduce the strength of the set.

Sub-Level Caving

A sub-level caving system is sometimes employed in mining the upper part of an orebody preparatory to top slicing.

The details of the system are shown in Fig. 6. Slices 40 ft. wide are cut up by drifts 20 ft. below the caved ground. The back of ore is mined by retreating and the waste allowed to cave. As no mat is used, the inevitable mixing of waste rock with ore causes a serious fall in the grade of the ore.



Longitudinal Section. Cross-Section A-A

FIG. 6.—SUB-LEVEL CAVING.

Rill Stoping

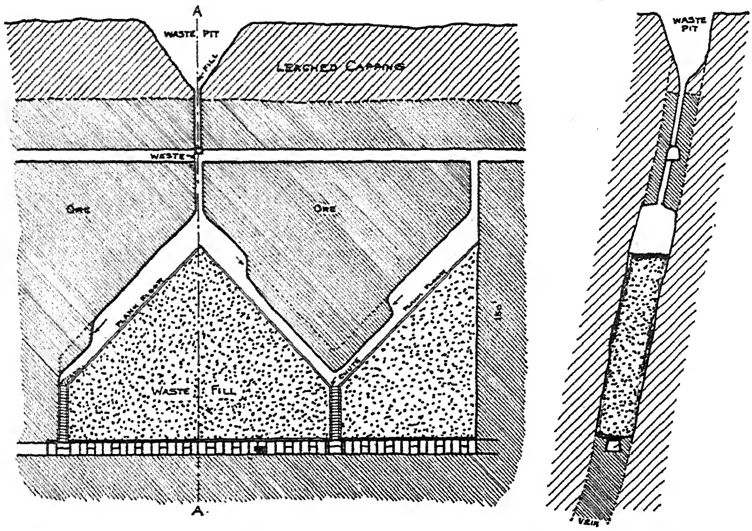
In narrow veins where filling is needed to support the walls, rill stoping is often employed. (Fig. 7.)

In this system the roof of the stope is worked on an incline and the broken ore runs down the sloping fill directly into the chute. After a slice of ore 10 to 15 ft. in height has been worked out, the stope is cleaned, waste filling run in and mining resumed.

Shoveling is eliminated, but the output of ore is necessarily intermittent.

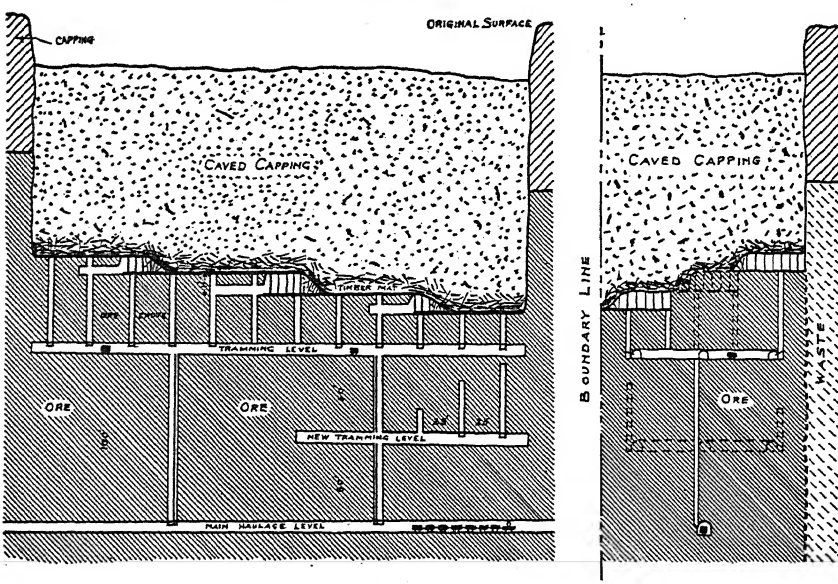
Top Slicing

The top-slicing method is principally used in the extraction of the large, soft ore orebodies of the Humboldt mine. The system is an appli-



Longitudinal Section. Cross-Section A-A.

FIG. 7.—RILL STOPING.



Longitudinal Section. End Section.

FIG. 8.—TOP SLICING.

cation of the longwall retreating of the coal mines, worked under an artificial roof of timber called the "mat." (Fig. 8.)

The top of the orebody is removed either by open stoping, square setting, or sub-level caving and a double floor of 2-in. plank laid. The overburden is then caved on top of the mat, either to surface or sufficiently high to make it safe to work beneath the mat.

A tramming level is laid out 55 ft. below the mat and divided by drifts into blocks of 30 to 40 ft. in width. At intervals of 25 ft. along these drifts, timbered raises divided into chutes and ladderways are carried up to the mat above. Commencing at the boundary of the ore, slices 11 ft. in height by 30 or 40 ft. in width are mined beneath the mat, which is temporarily supported by sets with posts 10 ft. high and round unframed caps. As the slice advances, a layer of 2-in. plank is laid on the floor, lengthwise with the slice, and the timber mat is caved by blasting the posts behind the working face; only sufficient space is kept open under the mat to permit of shoveling into the chute. Connection is made between the chutes by driving a tunnel ahead of the slice. This tunnel may be small and untimbered or carried the full height of the slice and timbered with 10-ft. sets to support the mat.

From the drift chutes, the ore is transferred by hand tramming in cars of 21 cu. ft. capacity to the motor chutes on the main haulage road.

Top slicing allows of a more regular and larger output from a working breast than square setting. The breaking and shoveling of the ore is, however, not so advantageous as in square sets; the working face not being undercut and the shoveling being done on a rough floor. The weight of mat is heavy but fairly constant and sudden caves are exceptional. Owing to the smaller working space that must be kept open, the consumption of timber is less than with square sets.

To allow the broken ore to be shoveled directly into the chutes at all times, the following modification has been made: When the working face has advanced so far that the ore cannot be shoveled directly into the chute, the men are transferred to the next chute, and a new breast is opened and worked on all sides. In slices 30 ft. wide and with chutes 25 ft. apart, this system always permits direct shoveling and avoids the use of wheelbarrows.

Panel Slicing

To eliminate the heavy expense of installing and maintaining chute raises every 25 ft. in the slices, a system whereby the ore will be loaded into cars directly in the slice is about to be tried out. In this system, which has been called "panel slicing," all of the working faces and all tramming will be on the same floor and the use of tramming levels at 55-ft. intervals will be abandoned. (Figs. 9, 10, and 11.)

The working level will be divided into slices by crosscuts every 40 ft. along the main road. The main road and the parts of the crosscuts

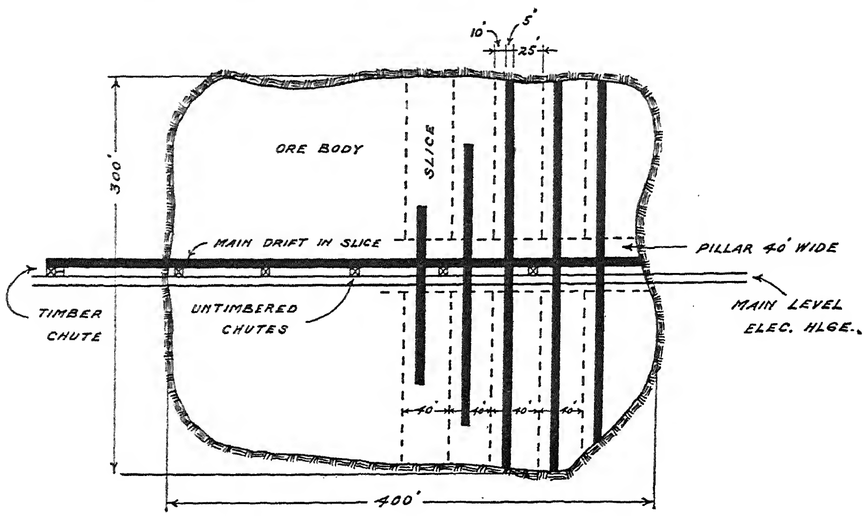


FIG. 9.—PANEL SLICING. PLAN OF PREPARATORY WORK.

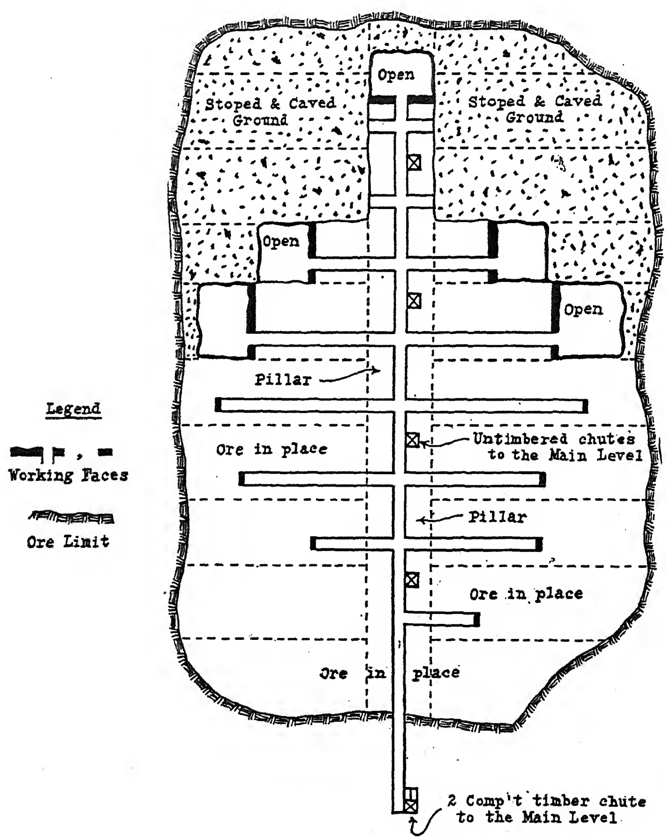


FIG. 10.—PANEL SLICING. PLAN OF WORKING FACES.

forming the main road pillar 40 ft. wide will be timbered with sets 6½ ft. high. Outside of this area, the crosscuts will be opened through to the mat above and timbered with slice sets 10 ft. high. Slices 40 ft. in width will then be worked from the margin of the ore toward the pillar over the roadway. This pillar will not be removed until the slices on each side of it are exhausted.

Ore chutes spaced at 80-ft. intervals along the main level receive the ore trammed from the working faces.

As one floor or panel is being worked, the next one, 11 ft. below, will be in preparation.

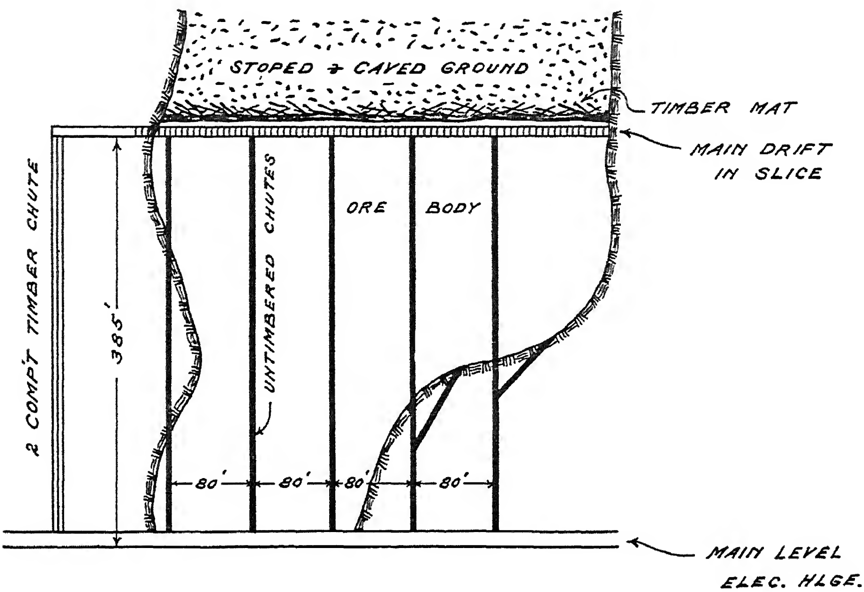


Fig. 11.—Panel Slicing. Section.

Drilling and Blasting

Air drills are employed almost entirely in stoping. Air at 100 lb. pressure is supplied by two Nordberg compressors of 4,000 cu. ft. total capacity, direct connected to two alternating-current motors. One compressor at Morenci supplies the Humboldt and Clay mines; the other at Coronado supplies the King and Coronado mines.

One-man machines of various makes are used throughout. In slicing, "jackhamers" are used in soft and water Leyners in hard ground.

Hercules ammonia powder of 30 to 40 per cent. is used in blasting, 1-lb. of powder breaking from 7 to 10 tons of ore. Prepared primers are issued to the miners from a number of supply magazines in the mine.

Timber and Timbering

Oregon fir costing about $24 per 1,000 ft. is used for all square timbers, round timbers being of Texas pine costing about $19. Round timber, being stronger and cheaper, is used wherever possible. In temporary timbering, such as in top slices, cheap cull lumber is used.

The standard sizes of timber sets are cut by a double-end framer at the terminus of the standard-gauge railroad at Clifton, before shipment to the mines.

Holman air hoists are used in handling the timbers to the stopes.

The dimensions of the timbers employed are as follows:

	Length Feet	Diameter Inches
Top-slice posts	10	7½ to 9
Top-slice caps	10	7½ to 9½
Top-slice flooring	10–15	2 by 12
Square-set posts	6½	8 to 12
	9½	8 to 12
Square-set caps	5	8 by 8
Square-set braces	4 7/12	6 by 8
Tunnel posts	6	10 to 12
Tunnel caps	5	11 to 13
Motor tunnel posts	7½	10 to 14
Motor tunnel caps	7½	12 to 16

The consumption of timber in top slicing, including chutes, ladders, and reinforcements, is about 9 ft. b.m. per ton, and in square-set work about 15 ft. b.m. per ton.

Mining Methods at Park City, Utah

BY JAMES HUMES,* PARK CITY, UTAH

(New York Meeting, February, 1915)

THE active mines in the Park City district at the present time are the Silver King Coalition, Daly-Judge, Daly West, and Silver King Consolidated. Several other companies, such as the Daly, American Flag, Ontario, etc., are doing considerable development work.

In the Silver King Coalition mines the ore is a replacement of a dark siliceous limestone bed, varying in thickness from 18 in. to several feet. In places the ore has reached a thickness of 25 ft. This limestone bed lies within 100 ft. of the underlying Weber quartzite. There are other lime beds, similar in appearance to this one, in the Park City formation, which is approximately 700 ft. thick; but not one of them has been found to contain any traces of ore. Directly under and in contact with the ore-bearing bed is another, composed of gray lime well studded with dark concretionary nodules; and this one is an unfailing indication of the proximity of the ore-bearing bed.

These ore deposits make out from or have connection with some one of the ore-bearing fissures, which were undoubtedly the channels for the circulation of mineral-bearing solutions. It is therefore proper to run prospecting drifts in the fissures, although they seldom contain much ore.

As a general rule, drifting in the fissures is not costly. Mining and mucking cost from $2.50 to $4 per foot, and very little timber is required. There are instances, as in the Daly fissure in the Daly-Judge mine, where the expense of keeping drifts open in the fissure is so great that it has proved economical to make the drifts in the foot-wall rock and run cross-cuts into the fissure at short intervals.

In the Park City mines it is not easy to determine beforehand the best method for extracting the ore from a newly discovered ore shoot in the bedded deposits. While we may get one dimension from the drift run in the fissure, that will not give us a correct idea as to the other dimensions of the orebody. The fissures here referred to have a N. 35° E.

* Superintendent of the Silver King Coalition Mines Co.

strike and the beds dip from 10° to 30° NW. The accompanying sketch map, Fig. 1, is a tracing showing part of a worked-out orebody, and has a connection with an ore-bearing fissure at the southeast end and 200 ft. higher up on the dip of the beds. It will be noticed that the 900 drift was advanced a long way in barren rock, yet it was all the time in the replacement bed, and where the ore is shown at the top of the incline it was very thin and discouraging, but persistent exploration developed a fine body of ore.

The orebodies are somewhat lenticular in form, with their greatest length along the strike of the lime beds. The perimeter, except on the fissure side, is very irregular in outline. There are numerous branches,

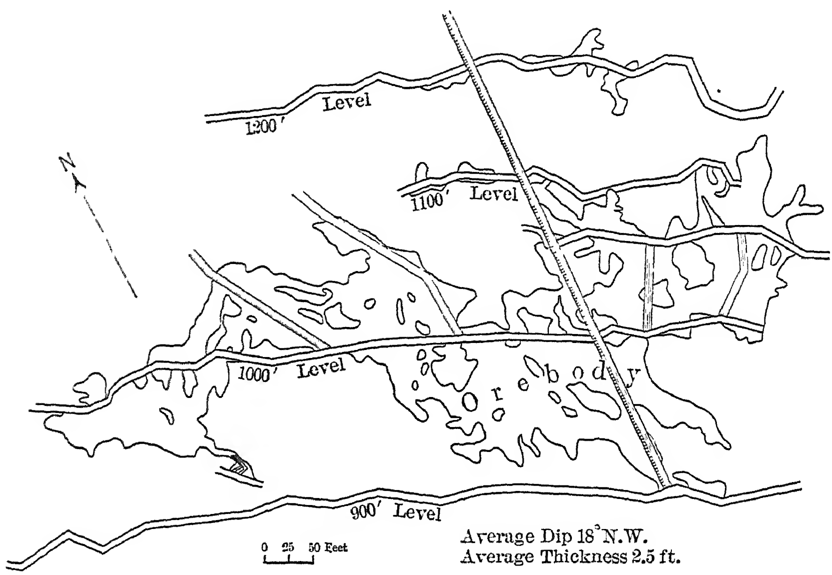


Fig. 1.—Sketch Map of Worked-Out Orebody.

extending sometimes as much as 100 ft., and averaging from 5 to 25 ft. in width. We also find the ore making up and down in small fissures for a few feet; and we encounter "rolls" in the foot-wall that make the beds almost vertical for from 5 to 25 ft.

Most frequently the ore dips down from the fissures (that is, the north-dipping fissures); and if our drift in the fissure has cut the ore we work down on it as far as we can by shoveling the rock out. If it persists in going down, we rig up one of our many small hoists, of which we have over a dozen, ranging in size from a Holman stretcher-bar hoist to a double-cylinder 35 h.p. These little hoists enable us to raise our regular mine cars. If the angle is too steep for a direct pull we simply fasten the end of the hoisting rope to a post at the top of the inclined winze, and pass it through a 7-in. pulley which is hooked into the car. This method gives

the engine a "double purchase;" and even with our little double-cylinder 5 by 6 in. engines we are seldom unable to manage our 18 cu. ft. cars. If the deposit is too thin, we take up enough of the bottom to admit the car. Very often our temporary inclines will meet with a face of barren rock. In such cases we do not cut through the waste but turn our rails to the right or left, as the case may be, and follow the ore. (See sketch map.) Should the ore continue down on the dip for a considerable distance, we run small drifts right and left from our main incline, in which we use a small special car, dumping its contents into a hopper that connects with the car on the main incline. Should there be a sufficient thickness of ore at the point where we wish to run one of these sublevels, we simply put down a track switch on the main incline, the points of which we throw with a lever on the level that is connected with the points by means of a long rod, crank, and short rod, and in this way we are able to load the mine cars that are hauled to the ore bins on the surface.

An experienced miner, reading these lines, will probably ask himself the question, "Why don't they do their development work from the lower edge of their orebodies?" As I said before, it is impossible to predetermine the outlines of such orebodies. When we took charge of this property we had many schemes for moving the ore from the faces to the levels. The best and easiest of application was the shaking launder, but as time went on we abandoned the thought of trying any of them. At the present time we do as much "underhand stoping" as overhead stoping, and in doing so we avoid doing a great deal of development work. Most of the time the deposit is so flat and thin that it is impossible to use a stoping machine to advantage, so that most of our mining is done by the "single jack" method. Sometimes the ore gets down to a few inches thick, but we do not abandon it until there is no sign of ore; for many times a small stringer has led us to a large deposit. The cost of mining ore here ranges from $1 to $4 per ton.

We believe that the above methods prevail in all the mines here where they are mining the replacement deposits. Following is a scale adopted by the Daly-Judge Mining Co. for running drifts and crosscuts with Leyner machines, the company furnishing all supplies:

Average Footage per Shift	Machine Men	Machine Helpers	Muckers
Less than 3.5	$3.25	$3.25	$3.00
3.5 to 4.0..........	3.50	3.25	3.00
4.0 to 4.5..........	3.75	3.50	3.00
4.5 to 5.0..........	4.00	3.50	3.00
5.0 to 6.0..........	4.25	3.75	3.25
6.0 to 7.0..........	4.50	4.00	3.25

Some Defects of the United States Mining Law

BY COURTENAY DE KALB, TUCSON, ARIZ.

(New York Meeting, February, 1915)

Revision of the United States mining law is needed chiefly because of the following reasons:

1. The conceptions as to the characteristics of orebodies that were held at the time the statute of 1872 was enacted have since proved to be in many respects erroneous.
2. The operation of the existing law has been found to work unnecessary hardship upon many claimants to mineral land.
3. The existing statute fails in large degree to encourage development of mineral resources.
4. The existing law often produces conflict between placer and lode-claim rights, working unnecessary injustice to placer claimants.
5. The methods of establishing the rights of locators on United States mineral land are defective, and are not uniform throughout the States within which such lands exist.

Law of the Apex

The objections to the so-called "law of the apex" are so well known as to require no detailed statement. It is sufficient to point out that the apex rule was based upon the notion that ore deposits consisted of well-formed veins included between definite walls, like the slice of ham in a sandwich. The fact is that veins conforming to that simple type are rare. Irregularity is the rule, and the chief metal production of the United States comes from masses which bear no resemblance to the tabular form of vein. Manifestly the law of the apex fails completely in such cases. Moreover, the tendency is constantly toward the exploitation of ores of lower grade than could formerly be worked, and such deposits exist mainly in the form of large irregular impregnated zones and masses, so that the future development of our mineral resources would be continuously embarrassed by shadowy claims to extra-lateral rights by the locators of so-called apexes.

Initiation of Mineral Rights

The spirit and intent of the United States mining statute possess the merit of liberality toward the poor prospector, in which respect our law is superior to that of most other countries. This spirit must be jealously maintained, and all efforts to enact a rich man's law must be resisted. The advantage to the nation through the actual development and exploitation of our mineral resources is vastly greater than the benefits represented by the sums that might be paid into the Federal treasury in the form of purchase price of claims, or as rental under a leasehold system. The development of industry means the broadening of opportunity for many workers and their families, provides room at home for an increasing population, and involves income to the government through taxation so large in the aggregate as to dwarf utterly the revenue that might be derived from the sale or rental of mineral lands. The leasehold system is peculiarly objectionable from this point of view, since it necessarily regards the mineral resources as a basis of direct taxation, payable as a condition of tenure, and this leads to a restriction of the right of prospectors to extract and ship metalliferous products without prepayment of the rental. It therefore imposes a burden upon locators from the beginning of their work, seriously limiting the opportunity for poor men to prove the value of their claims. The law should in every possible way promote the discovery and exploitation of the mineral resources, and should never prove a deterrent.

Under the existing statute the locator of a claim is supposed to have made an actual discovery of valuable mineral; he is required to so state in his location notice; and, very logically, following the plain language of the statute, he is required to demonstrate such discovery in the case of a claim located in a National Forest before acquiring any rights to operate thereon. As a matter of fact, the actual discovery of valuable mineral on the surface is rare. Even in past decades the prospector has usually been led to locate a mining claim on the strength of recognized indications which could be expected to lead to a valuable deposit only after the performance of a considerable amount of costly exploratory and development work. To-day the dependence of the prospector is more than ever upon indications, since the richer deposits, yielding workable ore at the surface, have been mostly discovered. We are entering a new era of the mining industry, in which reliance upon geologic indications must be increasingly great. The prospector is no longer actuated by hopes of striking a bonanza with his toe as he wanders over the hills. He has absorbed a certain amount of geological information from the army of trained men who are exploiting the mines of the country, and he possesses a keener appreciation of the salient features of ore formation and of the conditions that warrant belief in more deeply seated deposits. To require a

declaration of actual discovery is, more often than not, to turn the locator of a claim into a perjurer. The law should frankly provide for valid location based upon evidence which to the best knowledge and belief of the locator gives promise of leading to the development of a valuable deposit. He should then be accorded ample time within which to prove his claim.

The Mining District

The statute of 1872, following the customs of the early miners of the West, provides for the organization of mining districts, with power to create certain regulations, and to record locations. This primitive system served a useful purpose when the West was young and politically unorganized, but to-day it constitutes an anomaly, utterly superfluous, and fraught with dangerous possibilities. The great majority of mining districts exist to-day in name only, and for the most part are wholly disregarded by locators. Nevertheless, in many States no laws have been enacted, or upheld by the courts of appeal where they had been enacted, to provide for securing valid claims without their intervention, and such mining districts as are still kept alive by active organizations are mostly maintained by and in the interest of rascals for the purpose of defrauding men who make important discoveries within their limits. It is no longer difficult to reach the county seat to record location notices, nor to ascertain the State laws governing validity of locations, but it is often an almost hopeless task to find the district recorder, or even to find out whether there is such an official, and it is a still harder task to make sure that the regulations painfully dug out of musty old notebooks which had lain amid rubbish in the corner of a cabin in the hills, actually constitute all the regulations which the district organization may have enacted at various periods of its history. There are many crafty scoundrels, who might have made daring bandits a generation ago, who practice the safer villainy to-day of dominating a mining district, like a Spanish *cacique*, lying in wait to trap the unwary prospector or development company by artifices which the present mining law renders possible. That which was a beneficent provision under frontier conditions has become the tool of crooks in a civilized society. The mining district has served its function, and should now be abolished.

Titles

The American is wedded to land tenure by title. The sentiment in favor of title and against possession under leasehold is fully warranted. A title once held, though forfeit to the State for delinquent taxes, still clouds any future title that may issue, until a court title has been obtained in an action brought against the former owner, his heirs, or assigns. This means that, under conditions which might render it impossible for a

man to comply with all the requirements for retaining a leasehold, the owner of a mining title in fee simple has ample time to recover from his disabilities, whether of war, of poverty, or sickness, make good his delinquencies, and finally, if need be, throw his case upon the mercies of a court of equity. The difference between tenure by title and tenure by leasehold may be illustrated by the homely simile of a man hanging in a shaft by a slender rope which has just the requisite strength to sustain his weight. If a single strand is worn through on the rocky edge of the shaft mouth, he falls; but if he were hanging by a rope strong enough to sustain many times his weight one strand after another might be cut through and the man could still hang on. The mining law is for both the rich and the poor, but it should not be strong enough for the rich and too weak for the poor. The prospector who has more hope than cash wants a rope strong enough to hold fast to that hope through all the storms of adversity.

It has been pointed out that tenure by assessment work is open to grave abuse, as the law stands. Not only do men credit themselves with larger wages than they could possibly earn if working for others, so as to swell the account quickly to the $100 limit, but they falsify, and they do work that is easy, just to hold the claim, rather than work that is effective in the development of the alleged mineral deposit. The purpose of the law is and should constantly be toward the stimulus of actual development. Its two functions are to stimulate and to protect. The present statute has unfortunately lent itself in this particular to the retardation of industry, to the purposes of the "dog in the manger," and to those of the grafter. This is an evil that should be cured. Every operator knows how persistently clever schemers locate claims around a promising mine without hope of finding ore, but with the expectation that some time the expanding operations of the successful company will need the ground for a tailing dump, or a millsite, or for its growing camp. There is no limit to the length of time that a claim may be held by the mere performance of assessment work. The law requires that $500 worth of work be done before patent may issue. That is equivalent to five years' assessment work. It is often open to question what may or may not constitute $500 worth of work. This depends on many factors; but there can be less question about the granting of title upon the demonstration of the existence on the claim of commercially valuable mineral. If the claimant does not find commercially valuable mineral during a period of five or six years it is fair to assume that he never will find it, and such claimant should be incapacitated from securing patent to the same ground by relocation, or as a result of assignment to him from other locators, for a period longer by one year than the time within which a locator must proceed to patent or abandon.

It will be seen that the foregoing is distinctly in line with two objects

of a proper mining statute: viz., to foment the mineral industry, and to prevent the use of the law for purposes of graft. There should further be some provision for determining whether abandoned claims are to be again open to location, or whether they should be classified under another heading and offered for sale as public land. If so offered, the price should be enough higher than that required, after the demonstration of valuable mineral, for the acquisition of a mining title, as to prove an incentive to acquire by development of mineral rather than by purchase as open public land.

Placer vs. Lode Claims

The existing statute requires that placer claims be located in conformity with the land divisions where the area has been surveyed; that is, they must be described as, for example, the N. W. 1/4 of section so-and-so, or the N. 1/2 of the S. W. 1/4 of the S. W. 1/4 of section so-and-so, of township and range as per this or that meridian. Theoretically the system is admirable. If the land is surveyed, describe it thus, on the co-ordinate principle, whereby any one may put his finger on the precise square upon a map which represents the claim in question. Successful mining, however, depends not alone on theory; it depends very much indeed upon the practical hard facts one clashes with in the field. The hard fact in this case is that in the more rugged and barren parts of the West, where mines have an ungovernable propensity for secreting themselves, the population is scanty, the ranches are confined to widely scattered *arroyos* where a few acres of arable land exist, and where the conflict of neighbors has not led to checking out the surveys. In large part the survey contracts in the remoter regions were let with an eye to political advantage, and were treated by the contractors as a private graft. Here and there an area will be found that was honestly surveyed, where the corners were actually set, even far back from the more frequented trails and roadways, where witness monuments were really erected or witness trees blazed and inscribed; but these are the exceptions. Usually the corners were set only where failure to do so might be readily detected. Hence the discoverer of a placer deposit might hunt in vain for any sign or mark to guide him to a known co-ordinate. In order to ascertain his position on the map he must either run out the survey himself from some corner stone many miles away, or at large expense hire a surveyor and his corps of assistants to do it for him. The law never contemplated throwing such an unjust burden upon the locators of placer claims, but that is precisely the disability under which they labor. In revising our mining law we must do so in the light of facts. When a locator cannot, with a reasonable amount of effort, ascertain his co-ordinate position according to the survey of record, he should be permitted to claim and hold in accord with the actual area and position of his placer as referred

to easily recognizable landmarks. Even though it result in cutting land on the bias, and in playing tricks with the chessboard map in the Surveyor General's office, this disturbance of the idealistic harmony would be no greater than that produced by the old confirmed Spanish grants which frightfully disfigure Uncle Sam's co-ordinate symmetry in every part of the Southwest; nor would it deface the maps or upset the calculations of locators any worse than the lopsided "lots" which surveyors have been permitted to mark off as fillers along the township lines where the surveys refused to close. These examples show how vain it is to expect perfection, and since the co-ordinate system of land survey has in reality been chopped into a crazy quilt by force of circumstances, and as vast areas have been reported as surveyed which have never seen either compass or transit, it is wholly unwarranted to demand that the locator of a placer claim should pretend to define the position of his claim in the way that the present law requires.

Furthermore, it is quite irrational to require, as results under the statute, that a locator take up areas lying on high ground, of no possible value for placer mining, just because they happen to fall within the subdivisions of a section needed to include the valuable bottoms. It withdraws land which might better be left open for other uses, and it imposes upon the placer miner the burden of buying extensive acreage which he cannot use, when he proceeds to patent. This means, therefore, that common sense demands, even in the case of association claims, the much-maligned "shoe-string" form of placer location as the one that fits the facts of nature and of an imperfectly realized public-land survey.

Another evil to be cured in revision of the mining law is the absurd conflict between the lode and the placer miner. At present a placer locator can obtain no prior rights as against a subsequent locator of a cross lode. Here again the dual purpose of the law, to foment and to protect, should be kept in view. A placer mine by its very nature cannot be expected to have a long life, while a lode mine may continue producing for many generations. There is consequently no reason why a placer title should carry rights to mineral in place beneath the alluvial deposit. In many of the Eastern States, following the ancient theory that the metals belong to the sovereign, land titles do not carry an exclusive right in metalliferous veins which may be discovered at some future time. Precedent for such a distinction is abundant. In this view a lode title could be issued subject to pre-existing superficial rights obtained by placer title, terminating, so far as mineral rights are concerned, at rock in place. No further interference as a result of débris, right of way, water rights, and the like, could be anticipated than that which occurs under the conditions contemplated by the existing law, which admits of placer claims adjoining both side lines of a lode claim 600 ft. wide. It certainly is a manifest injustice to a placer locator that he be made subject to an ad-

verse lode location of later date, which may cut the placer claim in the middle, and seriously hinder its proper economic exploitation, or wholly preclude its operation. The common practice of locating supposed lode claims across a placer, for purposes of blackmail, is in itself a severe criticism upon the provisions of the existing statute, and is a sufficient warrant for careful revision in this respect. As matters now stand the locators of placer claims who hope to operate undisturbed by litigation from later lode claimants are forced to locate every cross seam, or prominent ledge of rock, or faulted zone, as lode claims, in order to protect their placer claims, thus incurring a great deal of unnecessary expense. That which embarrasses and heightens the cost of production is not in line with the theory of the conservation of natural resources.

Universality of the Law

Out of the same theory that provided for the establishment of local mining districts comes the opportunity for the several States to further complicate the mining law by legislation concerning the details of location, proof work, record, and other details. The failure of the Federal statute to settle all these matters was due to the frankly confessed ignorance of the majority of the representatives in the Congress, concerning mines, and ore deposits in particular, and concerning Western conditions in general. The congressional debates at the time, especially those in the Senate, where the bill was the subject of very earnest consideration, show the helplessness of the Congress in the face of these matters. There was then no Geological Survey with a vast store of facts and a corps of trained experts to guide and counsel; there was no Bureau of Mines to assist with testimony as to actual mining conditions; there were no trains of Pullman cars to facilitate the acquaintance of the Easterner with the West. Except for the good sense of the late Senator Stewart of Nevada, whose fund of practical information and forcefulness of character enabled him to dominate his colleagues, the law would not have been sound enough to have stood the test of 42 years as a fairly satisfactory instrument for the protection of an industry developing out of the relatively small beginnings of a frontier epoch into the mammoth enterprises which flourish under it to-day. The success of the mining law compels profound admiration and respect. Its deficiencies, however, as revealed by changing conditions and expanding knowledge, call for revision; and in curing the evils which more directly affect tenure in mineral land, it is also important to provide for those things which come under the head of "regulations," and which have hitherto fallen within the purview of mining districts and State legislatures, to the end that uniformity in law and procedure touching mining claims may exist wherever there remain public mineral lands open to location.

Discussion

Horace V. Winchell, Minneapolis, Minn.—Mr. DeKalb summarizes the reasons which he believes are important for the revision of the mining laws, as follows:

> "1. The conceptions as to the characteristics of orebodies that were held at the time the statute of 1872 was enacted have since proved to be in many respects erroneous."

We all appreciate the force of that suggestion. The idea of the old prospector who made rules and regulations in the early days, which were attempted to be incorporated, so far as possible, in the Mining Act of 1872, was that all of the veins in a certain district outcropped upon the surface, or approximately parallel to each other upon the surface, and maintained that parallelism beneath the surface. Therefore, it seemed perfectly proper and natural for each man to stake his claim upon an outcropping, and have the right to follow that vein in working downward, with the general expectation that his workings would always remain just as far away from those of his neighbor as the outcroppings upon the surface. This does not happen to be the case. Dr. Raymond has very ingeniously referred to the subject in one case, in words which read as follows:

> "If all mining properties presented this beautiful simplicity of structure (ideal location of ideal vein), and all mining locators exhibited a corresponding simplicity of purpose, the application of the law would be easy, but the naïveté of the statute fares badly between the freaks of nature and the tricks of man."

Mr. DeKalb further says:

> "2. The operation of the existing law has been found to work unnecessary hardship upon many claimants to mineral land."

The particular feature of the law which I think was in the mind of Mr. DeKalb in writing that paragraph was the requirement of a discovery before location. No other country, so far as I am aware, requires the prospector to make an actual discovery of mineral in place before staking out his claim. The more you think of it, the more absurd it becomes. A prospector is a man searching for mineral, for a vein, for a deposit. He wishes to find something to which he can later obtain title. He must necessarily be protected in his possession while he is searching for his vein, but under the present law he is a trespasser upon the public domain until he has found his vein. Twenty-five years ago it was perhaps an easy matter to make a discovery without any prospecting work. To-day it is exceedingly difficult. Twenty-five years ago there were no National Forests, there were no geological or mining experts appointed by the government roaming over the country and closely scrutinizing the prospective work of the attempted locator. To-day if a man makes a location within the limits of the National

Forest, within a few days a geological expert comes along and says to him, "What kind of a location have you? Let me see. If it is a lode claim, have you a vein in place? If it is a placer claim, have you rich enough material to pay for working?" If you have neither of these, then you have no location, "skidoo." A prospector must be assured of a reasonable possession of title while he is searching for his vein, as well as subsequently to its discovery. Under our present law he has no such right.

The third point made by Mr. DeKalb is as follows:

"3. The existing statute fails in large degree to encourage development of mineral resources."

This is linked up with the preceding point. So far as the law discourages the prospector by denying him the right to stake a claim wherever he desires to work, and there within the limits of his claim search for the vein or deposit, the prospector will not be so numerous as in the past, and will not do the work of pioneering which has been so valuable heretofore.

The fourth point made by Mr. DeKalb is:

"4. The existing law often produces conflict between placer and lode-claim rights, working unnecessary injustice to placer claimants."

This is particularly true, because of the defect in the law which makes it possible for a placer claimant, years after acquiring title, to be compelled to defend himself against innumerable contests and court cases on the part of lode claimants who come into the district after railroads have been built or reduction plants established, mines developed, and who find a vein crossing patented placer grants. A lode claimant may at any time institute a contest, declaring that the applicant for the placer claim had not at the time he made application for patent declared the existence of veins within his ground, and therefore was guilty of fraud, and had not paid the price per acre which the government exacts for lode claims. If the later applicant is successful, the result is under our law as at present constituted that he takes away from the placer owner property to which he had for 20 years supposed himself entitled, and becomes by reason of the success of his contest the owner of the veins which lie within the placer ground.

The fifth point brought forward by Mr. DeKalb is:

"5. The methods of establishing the rights of locators on United States mineral lands are defective, and are not uniform throughout the States within which such lands exist."

This point is discussed at some length in the paper, and it is true that the different States have somewhat different provisions regarding the rights of locators and the particular work required in perfecting a location.

R. B. Brinsmade, Puebla, Pue., Mexico (communication to the Secretary*).—I agree heartily with Mr. DeKalb's five reasons why the United States mining law should be revised. I desire to criticise only certain ideas expressed under the headings "Initiation of Mineral Rights," "Titles," and "Universality of the Law."

Whatever the "spirit and intent" of the mining law, I consider that 42 years of practice have shown that it subserves the interests of neither the poor prospector nor the average citizen.

Considering the prospector, it is clear that his fundamental need is not freedom from taxation, but *good* mineral ground *available* for prospecting. Even under the equitable laws of primitive placer mining, when each worker is only allowed to locate the gravel which he can personally handle, the first comers get the choice of locations and the late arrivals either get inferior claims or find none at all unoccupied. But when each first comer is allowed to locate a large area, merely for speculative or forestalling purposes, as Mr. DeKalb acknowledges is now possible under the existing mining law, the handicap of late arrivals becomes so excessive as to discourage their productive efforts altogether.

A poor prospector would far rather pay the areal tax demanded by Peru ($7.50 gold for metallic and $3.75 for non-metallic ground, yearly, per hectare) and get a chance to work on *likely* ground, than to be locked out from everything but *barren* or *remote* locations by the patented or fraudulently held claims of the United States. A rich operator can often stand the expense of buying out a forestaller or of boring for deeply buried orebodies, but the poor prospector must have cheap ground and easily found mineral or give up the ghost.

Mr. DeKalb first eulogizes mining activity, and then proceeds to eulogize the freehold title because it permits a claimholder to take advantage of dilatory courts and to hang on to his claim long after he should have lost it for failure to pay his State taxes. My observations in many countries have convinced me that where such delays favor one worthy, but unfortunate, prospector, they enable lawyers, real-estate operators, bartenders and similar "mining men" to exclude a score of prospectors from all the likely ground in a whole camp.

I cannot understand how Mr. DeKalb can have "profound admiration and respect" for a mining law whose "tenure by assessment work is open to grave abuses" and "has unfortunately lent itself in this particular to the retardation of industry, to the purposes of the 'dog in the manger,' and to those of the grafter." But, even supposing that these abuses can be corrected (as Mr. DeKalb hopes) for *unpatented* claims, the granting of the patent or freehold title he recommends would at once relieve the land-owner from any further pressure to insure his activity,

* Received May 7, 1915.

as, under the trivial land-value taxes prevailing in most States, he could then hold his ground idle indefinitely at slight expense.

Therefore it appears that the requisite activity of mineral claim-holders can only be *perpetually* insured by refusal to grant them anything but the leasehold title. And this is the practice of Latin Europe and America, working under the Napoleonic Code. To *properly* enforce the assessment-work requirement on claimholders would require a complete reorganization of the Federal land office, both methods and personnel. As such a revolution seems remote, I believe mining activity could be more feasibly assured by substituting for the annual assessment work an areal tax of $5 an acre for metallic and $2.50 an acre for non-metallic minerals, or slightly greater than the successful areal tax of Peru, in order to conform with the higher wage-scale of the United States.

While the leasehold system, with a sufficient areal tax to prevent forestalling and monopoly, would satisfy all the needs of the poor prospector, it would still fall far short of doing justice to the average citizen. Mr. DeKalb, in his objection to the *direct* taxation of mineral land, evidently overlooks the *dual* nature of property which I have elsewhere explained.[1] As by the English common law[2] the ownership of even freehold land is retained by the nation—for "fee simple" means a feud or lease from the sovereign political power—it is evident that no government has a right to dispose of its mineral land to *certain* citizens without considering the rights of the *balance* to its usufruct.

If a man had a coal bed on his farm, he would not consider himself benefited by its development and extraction unless he also got a payment for the value of the coal in the ground. Similarly, no miners, however poor and worthy, should be allowed to extract and sell the minerals of a nation without paying to the government—the representative of the whole people—the royalty value of their output. A mineral-land policy should insure justice in the disposal of the landed heritage of the average citizen, even if thereby development be retarded or certain lucky individuals fail to get rich quick from the unearned increment in rich mineral deposits.

However, a direct and just tax on mines can be levied so as both to stimulate development and increase the average gains of mine operators. To achieve this, it is merely necessary to tax only the net profit or true mineral royalty, which is computed by subtracting from the gross proceeds the operating expenses plus the interest and amortization of the *true* capital invested (cost of construction and development). By this system a large number of mines would pay less taxes than at present and the tax quota of the industry would fall chiefly on those properties working

[1] Our National Resources and Our Federal Government, *Trans.*, xliv, 633 to 640 (1912).

[2] "Landed Estate," in Blackstone's *Principles of Law.*

bonanza orebodies. The details of this taxing system will be shortly published in my new book entitled *Landlordism.*

E. D. Gardner, Missoula, Mont. (communication to the Secretary*).—In reference to Mr. Winchell's discussion of Courtenay DeKalb's paper on the revision of the mining laws: While agreeing with Mr. DeKalb and Mr. Winchell that there is a necessity for a revision of the mining laws, I wish to call attention to a misconception in Mr. Winchell's discussion of Mr. DeKalb's article regarding the present procedure concerning mining locations within the National Forests. Mr. Winchell states:

"A prospector is a man searching for mineral, for a vein, for a deposit. He wishes to find something to which he can later obtain title. He must necessarily be protected in his possession while he is searching for his vein, but under the present law he is a trespasser upon the public domain until he has found his vein. Twenty-five years ago it was perhaps an easy matter to make a discovery without any prospecting work. To-day it is exceedingly difficult. Twenty-five years ago there were no National Forests, there were no geological or mining experts appointed by the government roaming over the country and closely scrutinizing the prospective work of the attempted locator. To-day if a man makes a location within the limits of the National Forest, within a few days a geological expert comes along and says to him, 'What kind of a location have you? Let me see. If it is a lode claim, have you a vein in place? If it is a placer claim, have you rich enough material to pay for working?' If you have neither of these, then you have no location, 'skidoo.' A prospector must be assured of a reasonable possession of title while he is searching for his vein, as well as subsequently to its discovery. Under our present law he has no such right."

The government employs geological and mineral experts to make examinations of mining claims within the National Forests; but, unless the claims are actively interfering with the administration of the Forests, no examinations are undertaken until application for patent is made. The proportion of mining claims examined prior to application for patent is very small, probably less than 1 per cent. of the total number located. There are thousands of mineral locations within the National Forests of which no examination has been made by the Forest Service, and in all probability never will be except in cases where application for patent is made.

I discussed this matter in a paper entitled Mining Claims within the National Forests, presented at the Salt Lake meeting of the Institute.[3]

In many mining districts in the West there is some opposition to the practice of the government in having mining claims within the National Forests inspected before the issuance of patent. Many mining men feel that they are entitled to patent to mining claims, particularly in established mining districts, even if they have not made mineral discoveries on the individual claims. When such claims are contested they blame the

* Received June 7, 1915.

[3] *Trans.*, xlix, 408 (1914).

Forest Service, while it is obvious that the grievance is against the mining laws.

Congress has set aside a part of the public domain for a specific purpose—for the growing of timber, and the protection of the watersheds. The fact that any ground has thus been withdrawn is *prima facie* evidence that it has a definite value for the uses prescribed by Congress. Is it unreasonable for the government to require mining claimants to comply with the Federal law before giving them absolute title to a part of its domain which has been designated by Congress as having a presumptive value for the purposes for which it has been reserved?

A prospector or miner is in no way disturbed by government agents in the enjoyment of his rights within National Forests. No examination is made of any ground located for mining purposes unless, as stated, it interferes with the administration of the Forest. A claim may interfere with the administration of the Forest when it conflicts with areas leased to individuals under what is known as a Special Use Permit; when it is included within a tract from which the government has sold, or contemplates selling, the timber; or where the claim is so located as to control rights of way over which it is necessary to transport Forest products.

At the present time, in some parts of the country, the timber business within the National Forests is seriously interfered with by unscrupulous locators of ground under the mining laws. In all such cases, the government has a direct interest at stake, and an examination of the interfering mining claims is made to determine their validity.

The Hydro-Electric Development of the Peninsular Power Co.

BY CHARLES V. SEASTONE,* MADISON, WIS.

(New York Meeting, February, 1915)

Location

THE hydro-electric plant of the Peninsular Power Co. is located at what is commonly known as Lower Twin Falls on the Menominee River. This location is about 3½ miles north of the city of Iron Mountain, Mich. The principal points of delivery of electric energy from this plant are in the mining regions adjacent to Florence, Wis., and Iron River, Mich., and in the cities of Iron River and Iron Mountain. The hydro-electric plant at Twin Falls and the substation of the Iron Mountain Electric Light & Power Co. are connected by a 6,600-volt transmission line carried on steel towers and steel poles, the latter being used within the city limits. The plant at Twin Falls and the substation at Iron River, about 36 miles distant, are connected by a 66,000-volt, duplicate, three-phase transmission line, supported on steel towers. At Iron River, are located a 1,500-kw. auxiliary steam plant, and the substation, from which the current is distributed to the mines in this vicinity. A description of the auxiliary plant, main and customers' substations, and of the main and secondary transmission lines, follows later in this paper.

General Layout

The relative location of the dam, forebay, and power house will be understood by referring to Fig. 1. The dam is constructed on a ledge of solid rock which forms Lower Twin Falls. This rock ledge offers an excellent location in that the height of the dam, and therefore the amount of concrete, are thereby reduced, and also in that practically no pumping was required in unwatering the dam site. The maximum head provided by the dam is 44 ft., with an average working head of from 40 to 42 ft.

The other structures, including the substructure of the power house,

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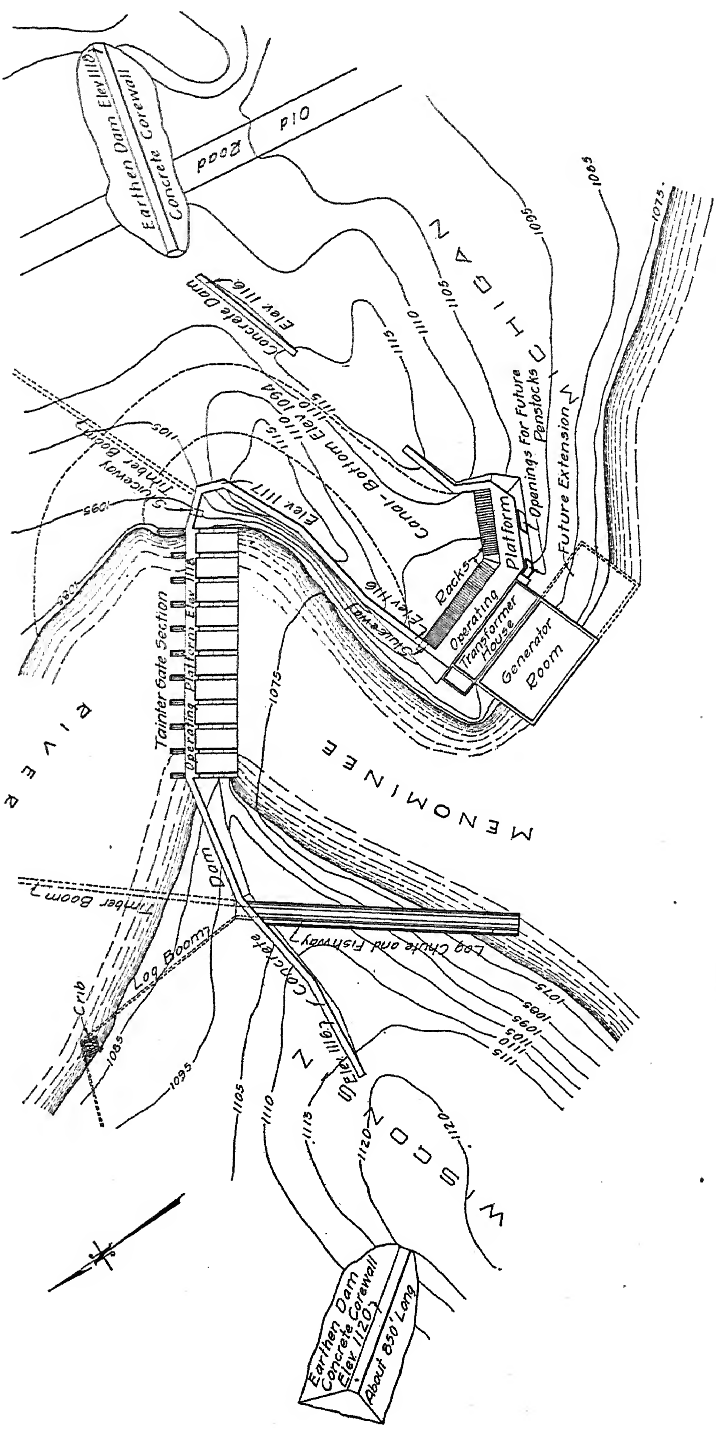


FIG. 1.—GENERAL PLAN OF THE TWIN FALLS PLANT OF THE PENINSULAR POWER CO.

the forebay or canal section through which the water is carried to the penstocks, and the head-gate section containing the trash racks and penstock gates, are founded on solid rock. The general plan of development shown in Fig. 1 was adopted, (1) because it afforded a location for the power house which is practically free from any serious effects

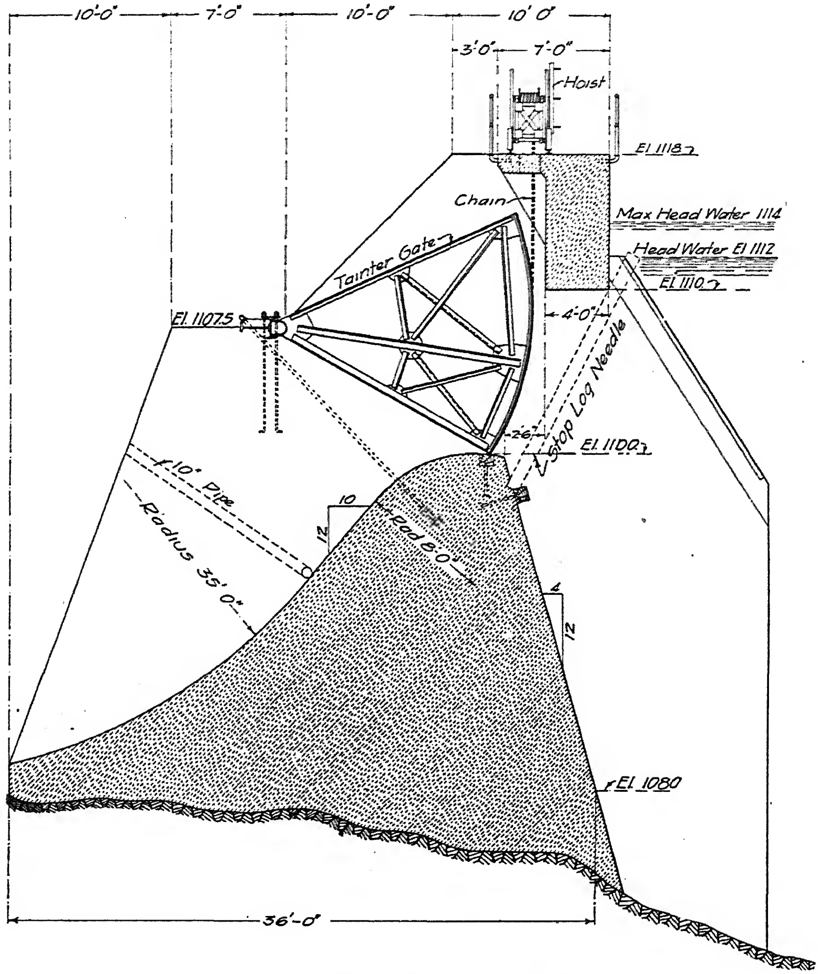


FIG. 2.—CROSS-SECTION OF DAM.

of floating materials, and (2) because of the ease of construction of the plant. The canal section is not lined except for a short distance where it joins the head-gate section, and is practically free from seepage.

Dam

The portion of the dam crossing the river proper consists of a Tainter-gate section for passing the flood flow of the river, consisting of 10

Tainter gates, each 14 ft. high and 14 ft. long. Fig. 2 is a cross-section through the Tainter-gate section and illustrates the general arrangement, design, and method of anchorage of the gates. This section is constructed of solid concrete throughout, and is of the gravity type, provided with a solid ogee section for the spillway. The Tainter- or flood-gate section of the dam, which is about 170 ft. long, is flanked on the Wisconsin side by a gravity-section concrete dam founded on solid rock. In addition to this, on the Wisconsin side there is an earthen section about 14 ft. high, and about 850 ft. long. This earthen section is 8 ft. wide on top with side slopes of 3 to 1 on the water side and 2 to 1 on the inside. This earthen section is provided with a solid concrete core wall, well anchored to the solid rock which underlies the surface. All of the rock in this vicinity is of a granitic formation, commonly known as greenstone. In addition to the flood gates, the dam is provided with a fishway of a type approved by the Wisconsin and the U. S. Fish Commissions. Next to the fishway is a chute to provide for the passage of logs or the floating of débris. Fig. 3 shows the general

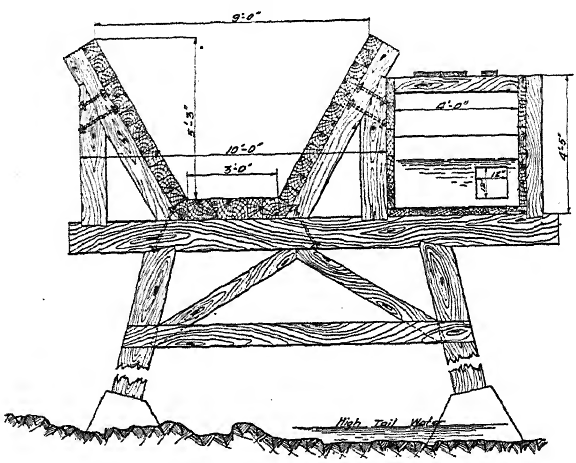


FIG. 3.—SECTION OF LOG SLUICE AND FISHWAY.

arrangement of the log chute and fishway and the general plan of construction. Suitable booms, anchored to heavy rock-filled cribs, extending to the bed of the river, are constructed above the dam for guiding the logs to the chute.

On the Michigan side, in the base of the Tainter-gate section, is a 4 by 4 ft. sluice gate, operated by a stand on the operating platform. This gate is utilized in the winter season when it is sometimes desirable to discharge only a small amount of water, and when, due to ice conditions, the operation of the Tainter gates is more or less inconvenient and difficult. The Tainter gates are operated by a hand winch which moves on a track on the operating platform.

Power House

The superstructure of the power house is constructed for the present installation of three 1,000-kw. generating units and the substructure has been completed for the two additional 1,000-kw. units which are to be installed in the near future. The head-gate section and canal or forebay leading thereto are also completed for the final installation of 5,000 kw. Figs. 4 and 5 show the general arrangement of the power house and head-gate section and the general arrangement of the switchboard, hydraulic, and electrical machinery.

The head gates are constructed of timber having steel bearing plates, and each is operated by a substantial individual hoist. Filler gates are provided in the penstock gates, although the hoists are designed to move

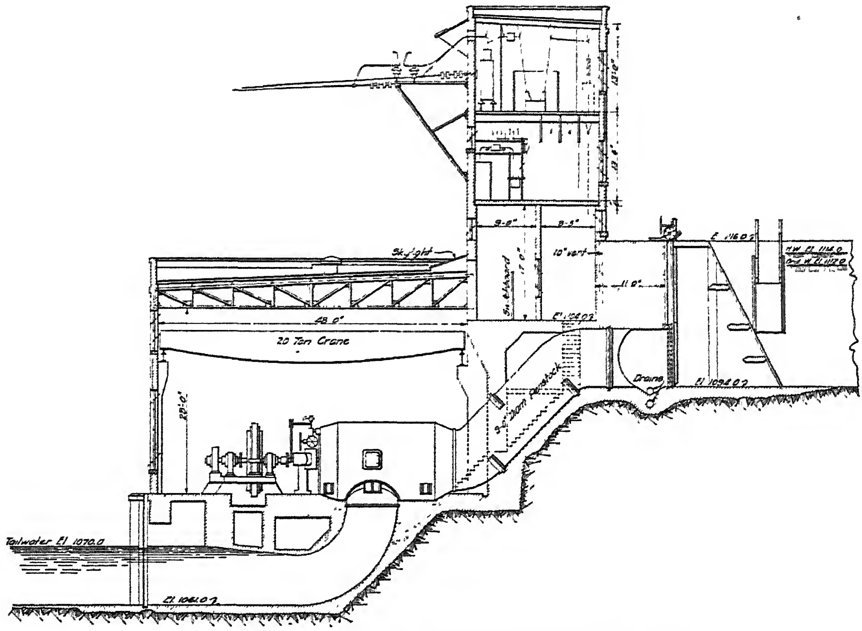


FIG. 4.—SECTIONAL ELEVATION, TWIN FALLS PLANT.

the latter without relieving the pressure. At the end of the trash racks is a sluice gate with a movable crest which provides means to sluice away ice or other débris that may collect in front of the racks. The latter are constructed from a special section, providing for a large entrance area, thereby reducing entrance losses and losses due to velocity head. The racks are so constructed as to admit of being easily raked and kept free from débris.

In the design of the hydraulic features of the plant, care was taken to provide liberal areas for waterways, and so arrange the plant as to minimize the labor necessary for its successful and economical operation.

The superstructure of the power house consists of a three-story brick building containing the switchboard gallery, individual transformer compartments provided with steel doors, high-tension switch compartments, etc., the general arrangement of which can be best seen by referring to Figs. 4 and 5.

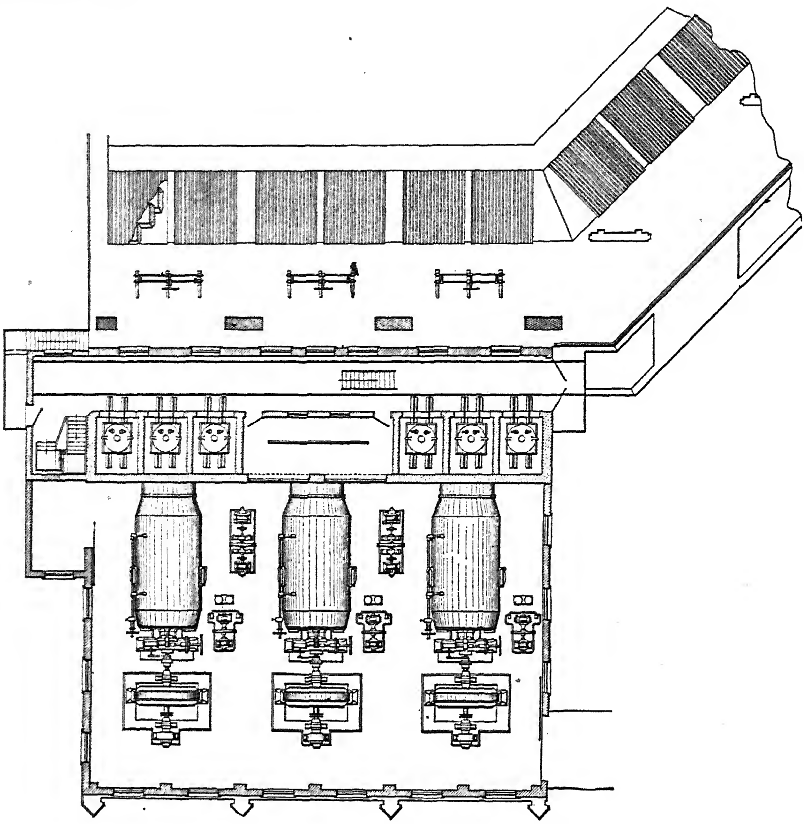


FIG. 5.—SECTIONAL PLAN, TWIN FALLS PLANT.

Stream Flow

The Menominee River forms part of the boundary between Wisconsin and the upper peninsula of Michigan. It is formed by the junction of the Brule and Michigamme Rivers, and flows in a southeasterly direction into Lake Michigan through Green Bay. A gauging station was established at the Homestead highway bridge about 2½ miles south of Iron Mountain by the U. S. Geological Survey, in September, 1902, and records are available from that time to date. Discharge measurements are made from the single span to which the gauge is attached. The rating of this section, that is, the relation between gauge heights and discharge, has been carefully checked, and the relationship as established

by the government has been found correct. The section of the river at this rating station is favorable for good results both as to straightness of channel and permanency of bottom.

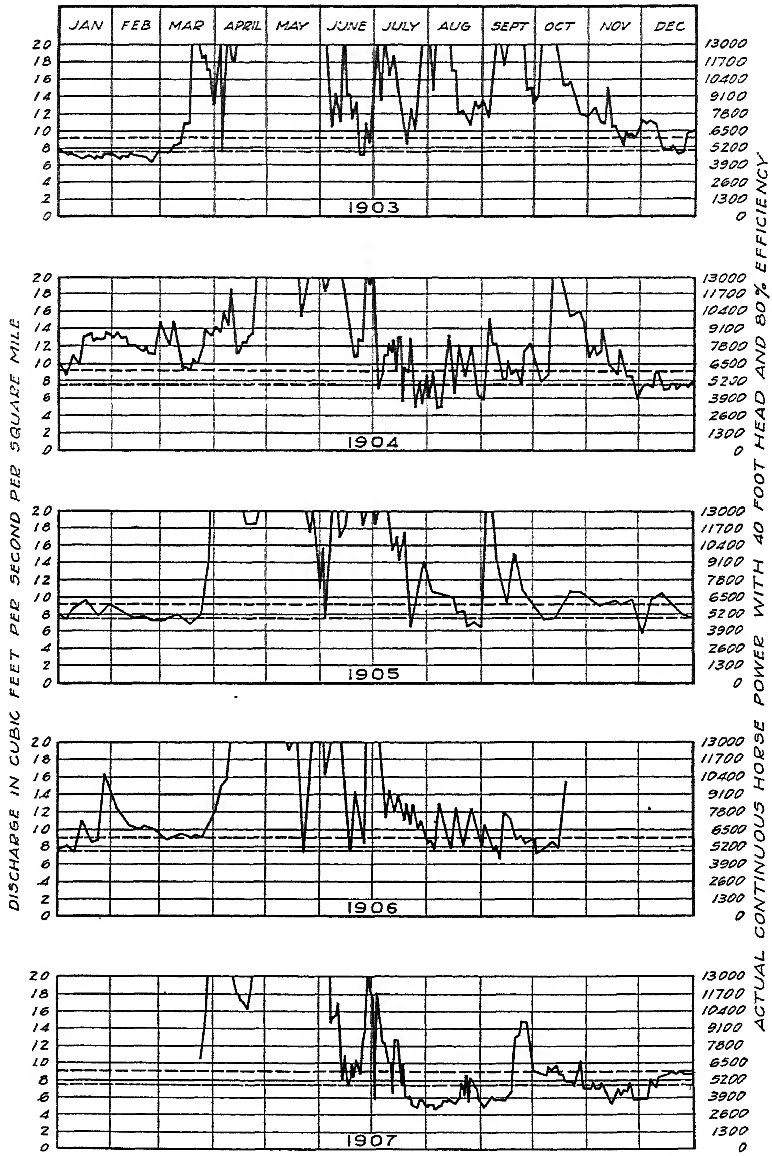
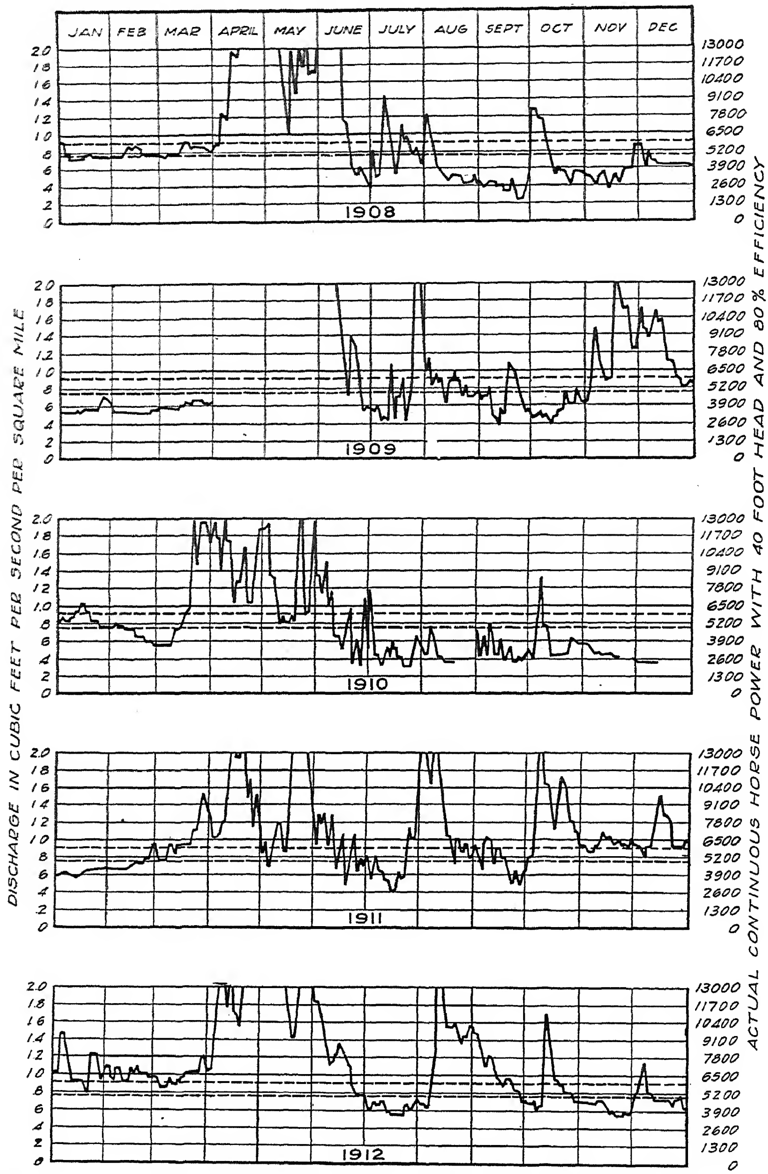


FIG. 6.—HYDROGRAPHS OF THE FLOW OF THE MENOMINEE RIVER, 1903 TO 1907.

Figs. 6 and 7 show hydrographs of the daily flow of the Menominee River for the years 1903 to 1912, inclusive, and are platted from the government gaugings referred to. The left-hand scale of these hydro-

graphs shows the discharge in cubic feet per second per square mile, which, if multiplied by the drainage area (1,790 square miles) above Twin Falls—the site of the hydraulic plant—and compared with the



[FIG. 7.—HYDROGRAPHS OF THE FLOW OF THE MENOMINEE RIVER, 1908 TO 1912.

irregular line of flow, will give the total discharge for any day of any year above mentioned. The right-hand scale shows the continuous horsepower at the turbine shaft with 40 ft. head and 80 per cent. efficiency.

This scale when compared with the irregular line of flow will give the horsepower at the turbine shaft for any day of any year for which records of flow are available. Inasmuch as it is possible to obtain a gross head of 44 ft. during periods of low flow, it is believed that the head used as a basis for calculating the amount of power will more than compensate for any losses that will occur. An examination of these hydrographs shows that, by the use of the pondage that is available above the dam, 2,500 continuous horsepower could have been delivered by the turbines during each day except for a short period during the years 1908, 1909, and 1910. This corresponds to about 1,500 continuous kilowatts delivered to the customer, allowing for proper losses in generators, transformers, and line.

With the 1,500-kw. steam-turbine plant at Iron River, the hydraulic output at Twin Falls can at low water be increased to about 5,000 h.p. The lower heavy line shown on each hydrograph represents the combined hydraulic and steam output with the present steam and hydraulic plant, and represents about 2,900 continuous kilowatts delivered to the customer.

There is also shown on the hydrographs an upper heavy horizontal line which corresponds to about 3,400 continuous kilowatts delivered to the customer. To deliver this continuous output and provide for the peak load would necessitate the full hydraulic equipment at Twin Falls of five 1,000-kw. units, one of which would constitute a reserve unit, together with a 2,000-kw. steam plant at Iron River, or 500 kw. in addition to that now installed.

Hydraulic Machinery

For the present installation, three units are provided. Each unit consists of a pair of 40-in. Samson horizontal-shaft, center-discharge turbines. Each unit is mounted in steel case penstock, and direct connected to a 1,250-kva. generator. Each turbine unit operating under an effective head of 42 ft. will develop 1,700 h.p. at full gate, running at a speed of 257 rev. per minute. With each turbine unit is installed a system of positive lubrication, consisting of a large grease compressor which is fitted with suitable hand wheel and gears for operating it. Holyoke tests of a similar 40-in. Samson runner showed almost 89 per cent. efficiency at about three-quarter gate, and the manufacturer's guarantee for the wheel installed was 86.5 per cent. at about three-quarter discharge —*i.e.*, when developing about 1,400 h.p. Under the conditions of installation it is believed that efficiencies somewhat greater than the manufacturer's guarantee will be realized.

For controlling the speed of these turbines three Lombard oil-pressure water-wheel governors are installed. These governors are of horizontal

design and are mounted on the bridge tree astride of the main turbine shaft close up to the flume head, thus occupying no floor space. With each of the governors is installed a motor-driven pump for 200-volt, three-phase, 60-cycle current. Individual pressure and vacuum tanks, also Lombard speed controllers for switchboard control, are likewise provided.

Generators

The generators are of Westinghouse make, and have a rating of 1,250 kva., 6,600 volts, 60 cycles, three phase, 257 rev. per minute, each equipped with a 120-volt direct-connected shunt-wound exciter of 27.5 kw. capacity, designed for operation with Tirrill regulator. The rotor of the generator is of the flywheel type, so designed as to assist materially in the speed regulation of the plant. All rotating parts are designed for 60 per cent. overspeed.

Transformers

The ultimate transformer installation will comprise six single-phase, 835-kva., oil-insulated, water-cooled transformers, arranged in two banks, delta-connected on the low-tension side, receiving 6,600 volts, and Y-connected on the high-tension side, delivering 66,000 volts. The present equipment consists of one bank of transformers with one extra single-phase unit acting as a spare.

Each transformer is placed on rails in a separate brick compartment closed by a steel rolling-curtain door. A transfer truck is provided to facilitate the moving of the transformers. In the west end of the transformer building an opening is left through all the floors large enough to allow the raising of the core out of the transformer with the aid of a triplex chain hoist attached to an I-beam on the upper floor. This opening was also used in hoisting material to the different floors during the construction of the plant. The cooling water for the transformers is supplied by two motor-driven centrifugal pumps, one of them being held for reserve.

A complete oil-piping system is installed, connecting each transformer with a Westinghouse oil-treating outfit of the filter-press type, so arranged as to permit cleaning the oil of any transformer without taking the transformer out of service.

Switchboard

The electrical energy generated is controlled by a complete switchboard. Fig. 8 is the wiring diagram for the generating station at Twin Falls. All oil switches are separately mounted and are controlled electrically from the main control board, located on the second floor between

the two transformer groups and overlooking the generator-room floor. All high-tension and low-tension measuring transformers are placed in fire-proof brick compartments, closed by ebony asbestos doors.

The individual busbar conductors of the high-tension, also of the low-tension system, are separated from each other by barriers of ebony asbestos wood. The 66,000-volt oil switches are of the horizontal break type.

Special care has been exercised in providing for an uninterrupted service, and the equipment installed for this purpose is unusually complete and may be classified as follows:

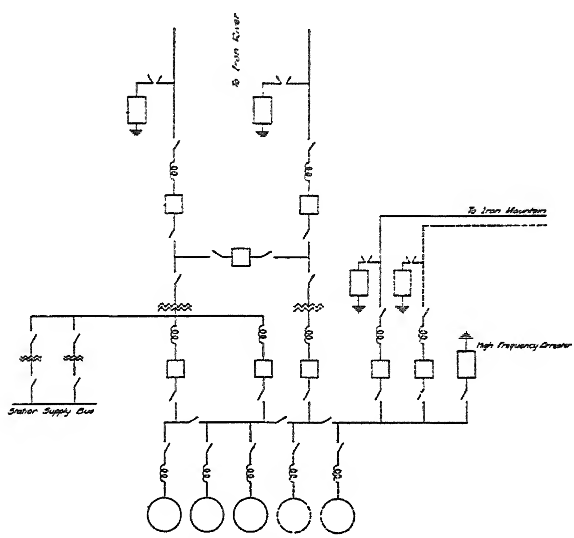


Fig. 8.—Wiring Diagram, Generating Station, Twin Falls.

a. Each three-phase line is equipped with a modern type of electrolytic lightning arrester.

b. In connection with the lightning arresters, each individual overhead conductor leaving the power house has been equipped with a choke coil, wound with Swedish iron to increase its resistance greatly under lightning oscillations.

c. All the high-tension station wiring has been done with solid Swedish iron, for the same reason.

d. All high-voltage current transformers have been protected by special electrolytic cells.

e. A special new type of arrester has been imported from Europe, in addition to the American types already installed, for the purpose of eliminating troubles due to high-frequency disturbances by lightning, switching, arcing grounds, or otherwise.

f. All circuits connecting with the busbars are provided with in-

dividual choke coils of Swedish iron to protect the machinery and station apparatus against high-frequency and short-circuit effects.

g. Two special motor-generator sets have been provided, each coupled to a heavy flywheel (laminated) of boiler-plate steel, to furnish emergency lighting and power to operate temporarily the oil switches in the very remote case of an accidental shutdown of the whole plant.

h. A special metallic grounding rheostat was installed between the transformer high-tension neutral and ground, together with a special current transformer, alarm bell, signal lamp, ammeter, and special relays. This equipment serves, in case of accidental ground on the transmission line, to limit the ground current to a moderate amount and keep the oil switches from opening automatically, thereby maintaining uninterrupted service notwithstanding trouble on the line. The rheostat is of such capacity as to give the station operator ample time to locate the line in trouble and either isolate same, or, more frequently in case of an arcing ground, to interrupt service over this one line for one second (maintaining service over the other line), and thereby remove the trouble.

Transmission Line

The transmission line leaves the power house through a large porcelain outlet bushing of the corrugated type, protected from the rain by a large concrete hood extending 5 ft. from the building wall. The line is built in duplicate and is supported on steel towers set in concrete bases. Two types of towers are used, namely, a flexible or two-post structure, the main members of which are channels, and a four-post tower constructed from angle sections, which serve as anchorages for the line. On all angles above 12° a heavier four-post structure is used. The percentages of the different types of towers are as follows: Standard two-post, 73 per cent.; standard four-post, 17 per cent.; angle towers, 9 per cent.; special towers, 1 per cent. Fig. 9 shows a standard two-post tower; Fig. 10, a standard four-post; Fig. 11, an angle tower; and Fig. 12 shows the special tower construction for the lines entering the substation at Iron River, Mich.

The concrete anchorages for the standard two- and four-post towers were constructed by molding concrete pedestals around the base or anchorage angles. The forms for these pedestals were constructed of galvanized iron and the work was done at convenient points along the line of construction, and the bases distributed to the site and set. The pedestals for the standard two- and four-post towers are 6 by 10 in. at the top, 9 by 11½ in. at a point 1 ft. from the bottom, and 18 by 24 in. at the bottom. They were cast 4 ft. 8 in. long, and the anchorage iron projects about 1 ft. above the top. Where a line parallels a railroad, and transportation is therefore facilitated, the method above outlined

for forming and setting the concrete anchorages has been found to be more economical than the usual method of depositing concrete in the earth around the anchorage iron, and also gives excellent results. For the heavy angle towers the last-named method was employed.

The conductors consist of six aluminum wires, of a total cross-section equivalent to No. 0 B & S. gauge, around a high-strength galvanized-steel center, having an elastic limit of 130,000 lb. per square inch. The power cables are strung with a sag considerably greater than that recommended

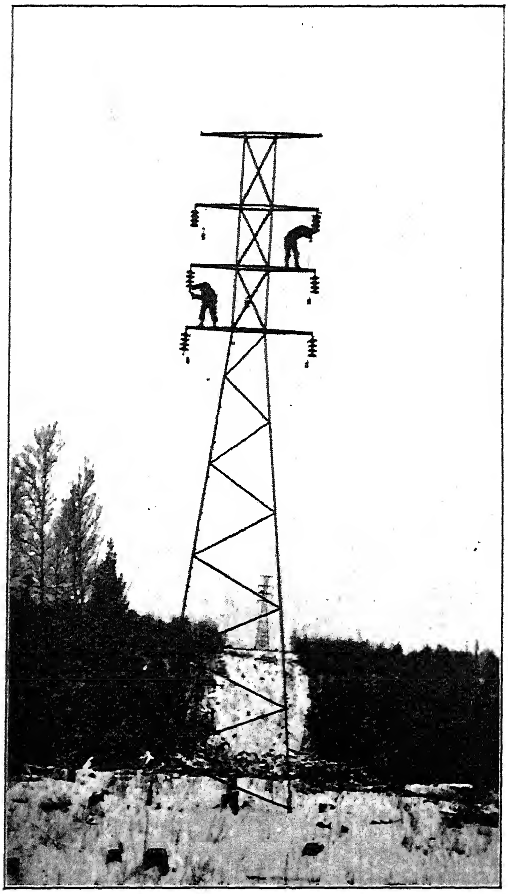


FIG. 9.—TWO-POST TOWER.

by the manufacturers and guaranteed by them as being entirely safe under the worst possible conditions of wind and sleet loads at low temperature. This was done to get a factor of safety on the line as high as could be practically obtained. The cables are supported from Locke porcelain suspension insulators, four units being used in series on suspensions and five units in series on dead ends.

Special care has also been taken in the design of this transmission line to provide for uninterrupted service under all conditions. The special features which will serve this purpose, in connection with those at the power plant mentioned under items *a* to *h*, are as follows:

i. The main line is built in duplicate, which always leaves the second line in service in case trouble develops on one line.

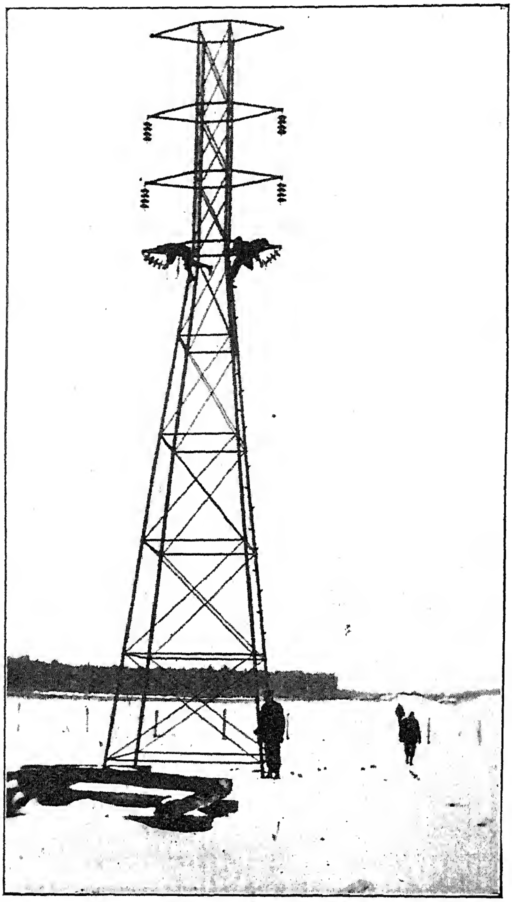


FIG. 10.—FOUR-POST TOWER.

j. The transmission line is protected over its entire distance by two overhead ground wires of No. 2 B. & S. copper-clad steel, strung 10 ft. 6 in. above the highest power cable.

k. Each transmission tower is thoroughly grounded by special ground plates below the concrete bases.

l. Each string of insulators is protected by special arcing rods arranged above and below the insulators to save them from damage by an accidental power arc.

m. The total line has been divided into a large number of separate sections, which may be easily and quickly disconnected by means of Dossert joints. This will greatly reduce the time for insulating and repairing a damaged section, if it should ever become necessary.

n. A private two-wire telephone line of No. 10 B. & S. copper-clad steel, supported on 6,000-volt insulators, has been installed on a wooden pole line, within a short distance of the transmission line, connecting the

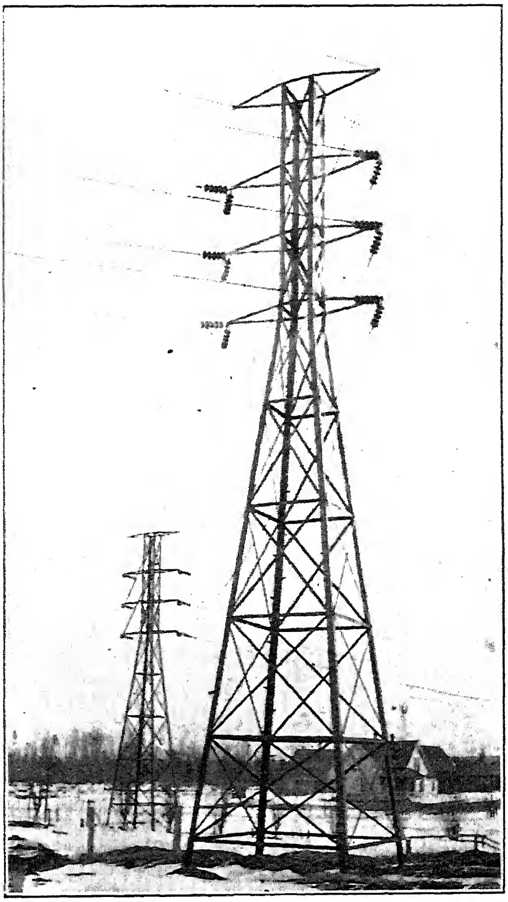


FIG. 11.—ANGLE TOWER.

main generating station with all the substations. The crews at the power house and all the substations are furthermore furnished with portable telephone sets which can be connected to any point of the telephone line. Each crew has also a portable acetylene search light at its disposal to enable it to patrol and repair the line at night.

With the above-named equipment, any trouble on the line can be quickly located, isolated, and repaired, considering also that the company

has at its disposal instant automobile service at all principal points along the line.

Auxiliary Steam Plant

The auxiliary steam plant is located just outside of the city limits of Iron River, which is central with respect to the present and possible future mining operations, and at the same time to the probable load center. The main line of the Chicago & Northwestern Railway through

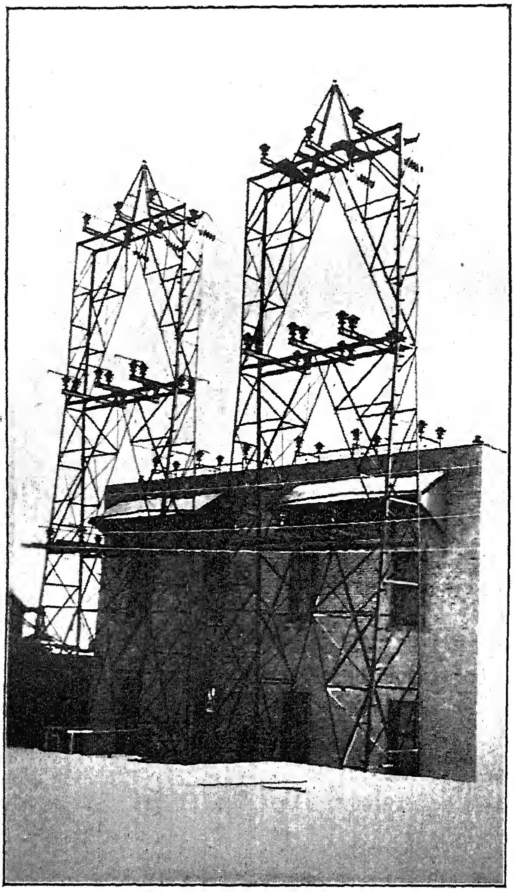


FIG. 12.—TOWERS AT IRON RIVER SUBSTATION.

this section adjoins the property on the east, providing excellent facilities for the delivery of coal. Just east of the railroad right-of-way is the Iron River, a stream having a drainage area of 60 square miles and an exceptionally uniform flow, thus assuring close at hand a sufficient supply of water for condensing and other purposes.

The permanent substation was completed first, and in connection therewith equipment, to provide enough steam capacity to take care of

power contracts in force before the completion of the hydraulic plant, was installed in a temporary wooden structure. Such portion of this steam equipment as was to be utilized in the permanent work was located on permanent foundations.

The first permanent installation, now completely finished and housed in concrete and brick buildings, consists of two Westinghouse turbo-generator units, one of 625 kw. and the other of 940 kw. capacity. The units generate three-phase, 60-cycle, 6,600-volt current. The turbines operate with 150 to 160 lb. steam pressure and 28-in. vacuum referred to sea level. On account of the elevation above sea level at Iron River, the indicated vacuum is about 27 in. of mercury.

The condensers are Westinghouse-LeBlanc with motor-driven pumps. Condensing water from the Iron River is taken from concrete wells located beside the condensers, the wells being connected with an intake in the Iron River by a 24-in. vitrified conduit.

For the initial steam-turbine installation, a boiler plant of three 150-h.p. long-furnace, internally fired boilers was provided, placed in a position out of the way of future boiler-plant construction. These boilers were selected on account of the facility of installation in the winter, very little bricking-up being required. They are provided with an extended breeching, connecting with a 60-in. by 75-ft. guyed steel stack set on a 25-ft. brick base, making the stack height 100 ft. above the grates.

The present permanent boiler equipment, now completed, consists of two 440-h.p. internally fired boilers, equipped with Hawley down-draft furnaces. These boilers are provided with an extended breeching connecting with a new reinforced-concrete chimney. The chimney has an inside diameter of 8 ft. at the top, and is about 154 ft. above the grates. The boilers first installed are still in their temporary boiler house and are connected to the new steam header for use as reserve.

In the boiler room two American steam pumps, one low service and one high service, are installed for boiler feed, together with one 1,800-h.p. Stillwell feed-water heater. The exhaust from the pumps and exciters is utilized in maintaining a feed-water temperature of 210° to 212°.

In the basement of the turbine room are the transformer oil-treating outfit, transformer cooling-water pumps, and general-service pumps, connected with an elevated tank in the boiler room for general station water supply.

The exciter units consist of one Westinghouse generator, driven by a vertical American Blower Co. engine, and one Westinghouse 10-kw. generator, driven by steam turbine. These units are on the main floor of the turbine room.

On the main floor of the turbine room, besides the main units and exciters, are the complete substation switchboard and duplicate sets of

15-kw. flywheel motor generators for control circuits and emergency lighting.

A 15-ton hand-operated traveling crane is provided for handling equipment in the turbine room.

The present plant is arranged and designed with a view to gaining high economy, and this is borne out by the results of operation during the first year. All coal used is weighed, which, with the records of electrical output, provides an index to the economy of operation. The stack temperature is obtained by an indicating and recording thermometer. A recording steam gauge and thermometer for reading all important steam, water, and oil temperatures is provided in addition to the usual steam and vacuum gauges.

Coal is dumped from cars, run out on a timber trestle, 100 ft. long and 20 ft. above the ground. This trestle is permanently constructed in a position parallel to the firing aisle and just outside the boiler room.

Main Substation

The main substation is at Iron River, Mich., and consists of a two-story brick building of fireproof construction, directly connecting with the company's auxiliary steam turbo-generator plant. It is arranged to receive two banks of transformers and the necessary switchboard and busbar equipment, all of which are now installed. There are six 420-kva. single-phase transformers of the oil-insulated water-cooled type. They are Y-connected on the high-tension side, receiving 66,000 volts between the leads, and delta-connected on the low-tension side, delivering 6,600 volts. Fig. 13 shows the wiring diagram of the substation.

The equipment for furnishing cooling water and that for filtering the transformer oil are duplicates of those installed at the main generating station. The substation is designed to control two incoming 66,000-volt transmission lines, and four outgoing 6,600-volt distribution feeders; also, two banks of station light and power transformers, and all the auxiliary electrical equipment installed in the substation and adjoining steam turbo-generator plant.

The same type of protective equipment as is installed in the main generating station has also been placed at the substation, such as: Electrolytic lightning arresters on all high-tension and low-tension circuits; individual choke coils on all high-tension and low-tension circuits; high-tension station wiring of Swedish iron; special electrolytic cells for protection of current transformers; high-frequency arrester of European make; flywheel motor-generator sets in duplicate. In addition, the 66,000-volt oil switches for the incoming duplicate transmission line are equipped with reverse power relays.

A second high-tension substation has been built at Florence, Wis.,

10 miles from the hydraulic plant, and connecting to the main 66,000-volt transmission line. This substation supplies power to two iron mines near Florence, over a 6,600-volt line carried on steel towers. It contains three single-phase, oil-insulated self-cooled transformers, each of 150-kva. rating. These transformers are delta-connected on the high-tension and low-tension sides to allow of uninterrupted service in case of damage to one transformer by operating the remaining two units on open delta.

A third high-tension substation has been built and is now in operation at Alpha, Mich., supplying power to the Judson and Balkan mines.

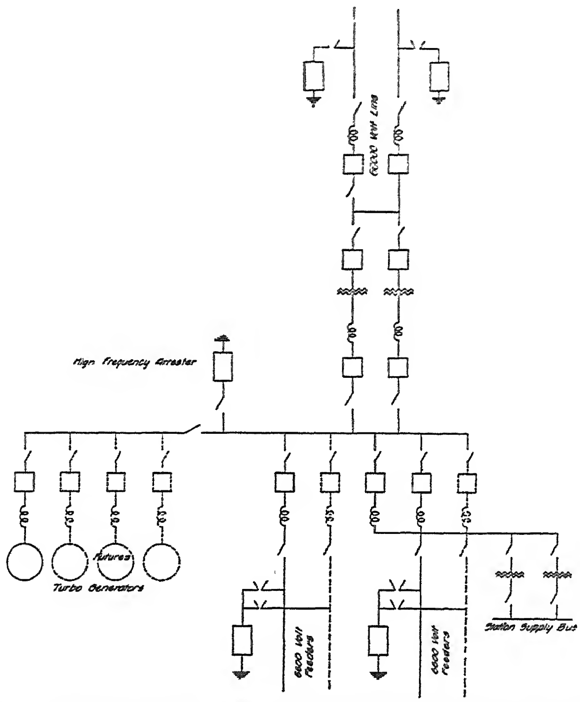


FIG. 13.—WIRING DIAGRAM, MAIN SUBSTATION AT IRON RIVER.

Arrangements are under way for furnishing the town of Alpha with electrical energy for general commercial lighting and power purposes. The equipment of this substation is the same as that at Florence, Wis., with the exception that the total transformer capacity amounts to 900 kw.

Secondary Distribution System

At present two three-phase, 6,600-volt feeder circuits are installed, distributing power to a number of mines around Iron River, Mich., and also furnishing energy to the local electric light and power plant. These feeders consist of the same size conductor as the main high-tension

transmission line (No. 0 B. & S. steel-reinforced aluminum strand), and are carried on 12,000-volt, pin-type insulators attached to steel towers set in concrete bases. The entire secondary distribution system, with the exception of one feeder which will be equipped later, is built in duplicate, thereby insuring uninterrupted service to the mines, even in the event of one circuit becoming totally disabled.

Fig. 14 illustrates the type of secondary tower that is used, the tower

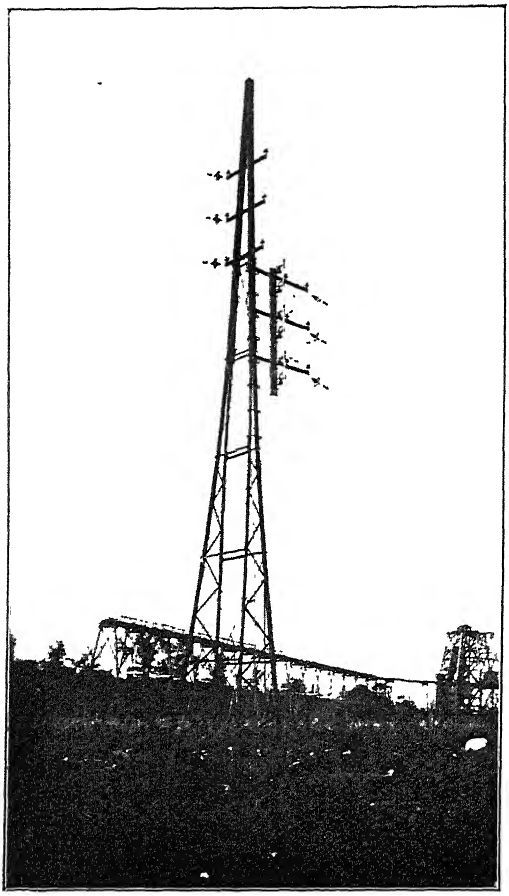


FIG. 14.—TOWER OF SECONDARY DISTRIBUTION SYSTEM.

in this photograph being a tap-off tower. The main members in these towers are channel sections, cut longitudinally for about two-thirds their length and spread to form the base width of 4 ft. 6 in. Each half of the tower was shipped to the field riveted up, leaving the bracing between the two halves to be done in the field. The base dimension at right angles to the line is 8 ft. 6 in. The same method of constructing the anchorages was used as heretofore described for the towers in the main line.

Like the main high-tension transmission line, the secondary feeder lines have also been equipped with protective devices to eliminate disturbances and maintain uninterrupted service. The special protective features embodied in these secondary distribution lines are as follows:

The line is protected over its entire distance by an overhead ground wire of No. 2 B. & S. gauge copper-clad steel, strung 7 ft. above the highest power wire.

Each tower is thoroughly grounded by special ground plates below the concrete bases.

Each insulator is protected by a special arcing rod to save it from damage by an accidental power arc.

Practically all the secondary circuits supplying the mines are built in duplicate.

Secondary Substations

Power is transmitted to each mine at 6,600 volts and is there transformed down to 2,200 volts for general distribution around the mine. The equipment of each of these secondary substations is owned by the power company and consists of three single-phase, oil-insulated, self-cooled transformers, of a capacity corresponding to the particular mine load, with ample reserve capacity for a considerable increase of this load in future.

The equipment further comprises an automatic oil switch, a control panel with all necessary instruments, relays and measuring transformers, and a complete set of electrolytic lightning arresters and choke coils. The transformers are delta-connected on both the 6,600-volt and 2,200-volt sides to provide for continuity of service in case of damage to one transformer, allowing the operation of the remaining two units in open delta.

The secondary substations carry a typical mining load, consisting of air compressors, pumps, hoists, motor generators for underground haulage, crushers, blowers, shop motors, and lighting above and below surface.

Practically all the larger motors, such as compressor, pump, and hoist motors, are operated on 2,200 volts, which calls for only a comparatively small investment on the part of the mine owner in step-down transformers for the lighting and small motor load, thereby maintaining at the same time a high all-around plant efficiency.

In order to obtain the best possible voltage regulation over the entire system, all the large air compressors are equipped with synchronous motors, so adjusted as to give a range of power factor most advantageous for the maintenance of a steady voltage. The maximum variation in voltage in the distribution system is about 4 per cent., which is well

inside of the limits generally allowed by public service commissions in distribution systems for city lighting.

The economic and other advantages of electric drive for mining operations have been fully realized in the district served by the power company, which is evidenced not only by the number of new mines with electrical equipment throughout, but also by the mines formerly operated by steam and now changing to electric drive. The economy effected by electric over steam drive is more evident on the pumping operations. Three different companies supplied by electric power from this plant report savings of from 30 to 70 per cent. in the power cost for pumping.

A special arrangement has been made by the power company in regard to the large hoists, with their intermittent short-time heavy loads, which had an unfavorable influence upon the total load factor of the electric mine installation, on which the rates for service are partly based. The new arrangement provides that all the large hoisting motors be supplied over a separate set of meters, which eliminates the effect of the intermittent heavy hoisting load upon the meter determining the load factor, and thereby reduces the rate for power which would otherwise result.

The power taken by the hoist, and also the other motors, is not charged on the basis of the actual load factor obtained at the installation, but on the more favorable basis of a load factor obtained by that portion of the load exclusive of the hoist.

The writer is indebted to M. H. Collbohm, electrical engineer, and other members of the office force, for valuable assistance in the preparation of this paper.

Safety Methods and Organization of United States Coal & Coke Co.

BY HOWARD N. EAVENSON, GARY, W. VA.

(New York Meeting, February, 1915)

THE mines of the United States Coal & Coke Co. are located in the Pocahontas coal field, in McDowell County, West Virginia. Twelve plants have been opened and equipped, of which, by reason of the present business depression, only nine are now in operation. These twelve plants serve eighteen mines; some of the plants having several openings, in either the same seam or in different seams, tributary to the same tipple. Only nine of these mines are now being worked at the plants in operation.

Construction work was begun in 1902, and coal from the first three plants started was shipped in December of that year. An additional plant was opened in 1903, two in 1904, two in 1905, two in 1907 and the last two in 1908. On Jan. 1, 1915, the plants have been in operation an average period of 8.8 years.

The rules of the company, which have been in operation since work was first begun, are similar to those in use by the H. C. Frick Coke Co. at its mines in Pennsylvania, in which "Safety First" was incorporated as early as 1907; and it has always been the policy of both companies to look carefully after the safety of their employees. Although the field is an extension of the old Pocahontas field, the mines, located upon a new branch railroad, are away by themselves, and, as is usual in such cases, the class of labor attracted at first was temporary in character. The rapid development of the mines required the employment of many new men and officials each year; and, while the accident record from 1904 to 1909 was as good as the average of the field, it was felt to be not as good as it could be made, and a determined campaign was inaugurated for preventing accidents of all kinds. Fig. 1 shows the tonnage of coal produced per fatality, both total and for inside accidents only, at our mines; the tonnage per fatality of all West Virginia mines, and of those in the Pocahontas field.

The purpose of this paper is not one of congratulation on the results so far accomplished, since the operating officials are not at all satisfied that the maximum is in sight, to say nothing of its having been reached; but to describe the organization and methods that have produced these

results, with the hope that this information may be of service to others interested in accident prevention. Fortunately, the widespread safety movement which has been agitating the entire country for the past six years has awakened interest in this subject; and it is a very unprogressive company that is not now doing all in its power to safeguard its employees in every way. The time has now arrived when as much attention should be paid to serious accidents—those causing a loss of time of 35 days or more or a permanent injury—as to fatal accidents. In many

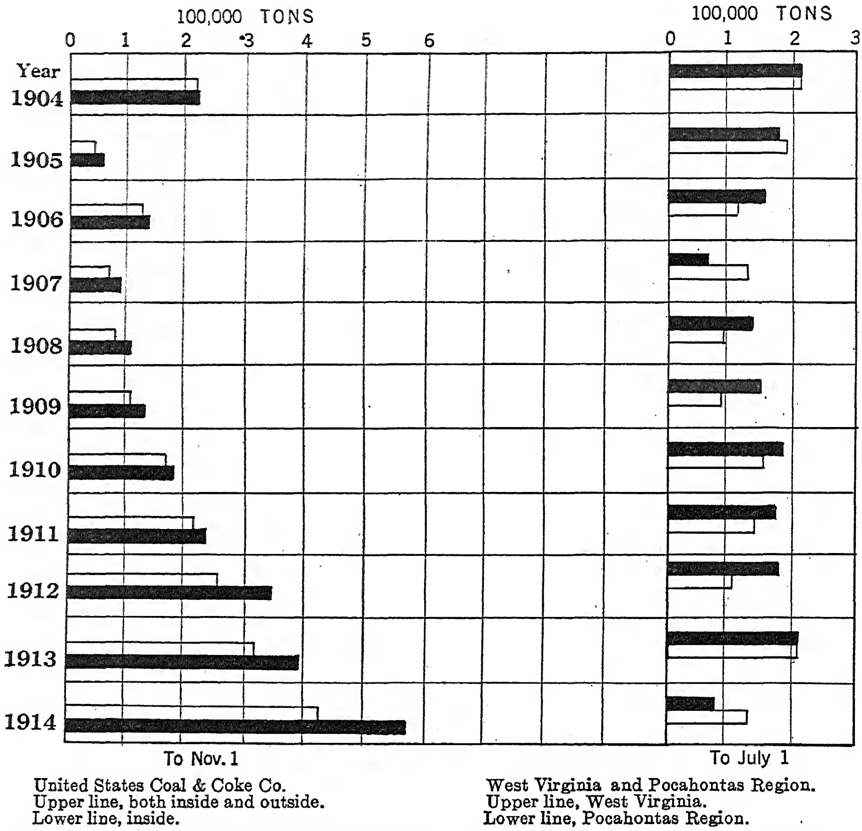


FIG. 1.—TONNAGE PRODUCED PER FATALITY IN THE MINES OF THE UNITED STATES COAL & COKE CO. AND IN WEST VIRGINIA AND THE POCAHONTAS REGION.

cases, the cause producing a serious accident would have produced a fatal one, with a few inches or a few seconds leeway; the difference being mainly a question of luck.

To those who have investigated mine accidents, the usual statement of the boss in charge of the mine that "he had been in the place an hour or so before the accident happened and had told the victim to set a post, but he waited to load another car before doing it," is entirely familiar; and,

while this is undoubtedly often the case and no reflection is intended on the hard-worked mine foremen who make it, it reveals a condition which must be removed before success in accident prevention can be expected. In our opinion, the first essential to success in work of this sort is the recognition of the fact that it is the duty of the employer, as far as possible, and by every means in his power, to prevent injury to the employee. So long as the view prevails that when the employee has been notified of any dangerous condition the management's responsibility is ended, effective prevention of accidents cannot be achieved.

Early in 1909 our inside organization was increased by adding to the list of officials at each mine assistant mine foremen enough to provide one for each 25 men employed. To each assistant was assigned a definite territory, over which he had the same authority that the mine foreman exercised over the entire mine. Where the territory assigned to an assistant was very large, the limit to the number of men in his charge was fixed by the number of places that he could inspect easily in 3 hr. The average number of men under each assistant foreman is about 22.

The fundamental idea in appointing these men was to provide enough executive officials at each mine to allow each one to stay in any working place where any dangerous condition was found until that condition was removed; and this is the one thing rigidly required of these men. Since this system was started, several accidents have happened, because this instruction was not strictly carried out; and it is now believed, with the experience of practically five years, that if this rule is rigidly followed, and the remaining rules of the company complied with, at least three-fourths of the accidents now happening can be prevented.

To acquaint these men thoroughly with the instructions of our higher officials and with the policy of the company in its mine work, complete projections are made, showing the proposed workings of each mine for a period of two or three years in advance; the station at which each place is turned and its course; the weight of track to be laid; the widths of the various places; the size of trolley wire to be used; the area on top of overcasts; whether overcasts and stoppings are to be permanent or temporary; and various other details, so that the future work to be done can be thoroughly understood by every official in the mine. In addition to this, a synopsis has been made in the form of a blue-print book of the instructions issued from time to time through circular letters by the General Superintendent. A copy of this book is given as Appendix A. It contains, under appropriate headings, in language that can readily be understood, the standards of the company for ventilation and inspection; explosives and shooting; track work, line sights, etc.; timbering, slate work, etc.; and general rules and standards for electric work. With these are bound standard plans for room timbering and track work, for various widths of places and depths of cuts, room switches, clearances and supports for trolley wires,

track bonding around switches, placing trolley frogs, etc. Each assistant mine foreman is furnished with a copy, and is required to familiarize himself with its contents.

After such additional supervision had been furnished for the inside workings of the company, a system of systematic timbering was introduced, under which every room in the mine is timbered, in accordance with the standard plan referred to above, unless the roof in the place is exceedingly bad and requires more than standard timbering. In other words, no matter how good the roof may be in any particular place, the standard amount of timber is required to be set there, and the only exception to this rule is that if bad conditions develop, additional timbers must be set to make the place safe. It was found that this was the only way in which the working places could be properly timbered. If the amount of timber to be set is left to the judgment of the miners or of the assistant mine foremen a loophole is provided for accidents and for excuses in case of accidents. The systematic timbering is designed to be sufficient for all except the very worst conditions that are found, and in these places additional precautions must be taken. This systematic timbering has been a large factor in the reduction of accidents.

It is an important duty of the assistant mine foreman to see that the timbers called for are set as soon as the coal is removed sufficiently to allow their placing, and not to allow the miners to wait until the day's loading is completed before setting any posts. The assistant mine foreman must see that all timbers that should be set in any place are so set before he leaves the place.

All shooting is done by the assistant mine foreman by electric battery; the miners drilling, loading, and tamping the holes with clay, which is furnished in each working place. The charge limit is 2 lb. of explosive per hole and only one hole is shot in any place at one time. None but permissible explosives are used, and all coal is undercut before being shot. About 85 per cent. of the cutting is done by machines; pick work is done only in room pillars, and in places where the weight renders machine cutting inadvisable or unnecessary. When the miner is ready for a shot, the assistant mine foreman comes to the place, sees that all "bug dust" has been loaded; that the place is clean; and the hole properly loaded; and that no coal dust has been used in the tamping. He then attaches the cable to the exploder wires and, after the miners and himself have retired to a safe place, fires the shot. After this, he inspects the face and roof, instructs the miners about pulling down loose coal or slate and setting posts, and *sees that this is done before he leaves.* Where pick mining is done, he sees that the place has been undercut before it is shot and that no shooting in solid coal is done.

On all haulage roads, and wherever more than one car is hauled at a time, 2.5 ft. clearance is provided on each side of the mine car. This

SAFETY THE FIRST CONSIDERATION

UNITED STATES COAL & COKE COMPANY

REPORT OF................MINE ....191..

MR. E. O'TOOLE
General Superintendent
Dear Sir:–
I visited...........Places, No. Men..........on......Section
and found it as follows:
1. Quantity of Air in last Cross Cut.............................
Timbering (State condition and numbers of places)...... ....
Clay Ballast on Haulage Roads.................................
Frogs, Switches and Derails not protected.........................
Bad Slate in Rooms and Ribs...................................
Bad Slate on Headings...
Powder Caps...........Tamping for Shots..................
Dangerous Practices with Stock.................................
Pieler and Safety Lamps..............Foreman's Cane.............
Danger Signs...........Letters Posted Regarding Accidents.......
Condition of Mine Cars..
2. Dirty Coal...
Slate and Rash Piles...
3. Cutting—Narrow..................Shallow..................
Timber Drawn...
Condition of Roads.. ......
Defective Wiring...
Line Sights....................Light Cars....................
Condition of Live Stock...
Feed and Water given Stock.....................................
Injured Stock in Stable from This Section........................
Mining Machines out for Repair.......Safety Mottoes in Place.....
Foremen's Record Book..........Material covered up or lost......
Condition of Pumps and Motors..................................
Miscellaneous...
....
....
....
(When anything is wrong, give number of place, etc.)
MERITS?............... ....
Inspector

covers all places in the mine except rooms. On all haulage roads, at intervals of not more than 175 ft., are placed electric lights, which not only allow the trolley wire to be plainly seen but have led to a marked improvement in the quality of the haulage track.

Trolley wires are protected by hanging boards at all places where men or mules cross under them. The current used is of 275 volts.

All officials—Superintendent, Mining Engineer, Mine Inspector, foremen and their assistants—are instructed not to leave any place in which any dangerous condition is found until that condition has been removed.

All new men employed inside are instructed in their duties by the Mine Inspector, and this fact, with their names, is noted on his report. The Mine Inspector and Mining Engineer also measure the amount of air in circulation and report the quantity passing the last break-through of each pair of headings; 12,000 cu. ft. per minute is the amount required by the regulations. The form used by the Inspector is shown on the preceding page.

Both inside and outside, the careful guarding of machinery was inaugurated. After an experience of two or three years and the injury of several men in machines thought to be well guarded, it was found better to design the guards so that it would be difficult to get at the machinery, even to oil it, rather than to make this work easy and run the possible chance of an accident. In other words, it was found necessary to make access to all machinery a matter of some trouble, even to those maintaining it.

The class of men available for assistant mine foremen was not, at first, as good as was desired. The problem, as in all accident-prevention work, was one of education; and it was found somewhat difficult to convince some of the men, particularly the older ones, that the company actually meant "safety" to be the first consideration. In fact, at the annual meeting held more than a year after this policy was first insisted upon one of our oldest mine foremen declared that he really did then for the first time believe that the company meant what it said in this matter.

In order to stimulate the interest of the assistant foremen in their work and to make the prevention of accidents a personal matter with them, a premium system was started in May, 1910, and has been, with some modifications found necessary, in effect since that time. Under this system, an assistant mine foreman having a clear accident record for a month receives a bonus of $5; should his record be free from accidents for six consecutive months, he receives a special premium of $10 per month in addition to the $5 already mentioned; and this bonus of $15 per month is paid him as long as his record remains clear. Should he have an accident he is charged with demerits for each man who is injured under his charge each month at the rate of 10 demerits for each minor, 20 demerits for each serious and 40 demerits for each fatal accident. No person having 10 or more demerits to his discredit at the end of each month is entitled

to any premium, but if he has less than 10 demerits he will receive the same premium as before. In addition to the accident record, no person is eligible for a premium for any month in any position who has not worked in that position every working day but one during the month, unless he shall have been promoted during the month from one position to another and is eligible in both positions. The intention of this requirement is to make the foremen work regularly. A number of accidents have occurred while the regular foreman was not working and his substitute was not equally familiar with the conditions. The work of every man must be satisfactory to his immediate superior. If it is not so, the superior has the right to charge him with demerits at the rate of 10 per month. This is a disciplinary measure, since it was found that a few foremen did not take sufficient interest in the prevention of accidents to attend the weekly meetings of the officials to discuss and investigate accidents which had occurred. But this provision has been used very seldom. Any assistant foreman in whose section the company's Mine Inspector finds any dangerous practices or dangerous conditions which might cause accidents, is charged with five demerits for each visit at which such conditions are found. This provision is required because an assistant mine foreman might permit dangerous conditions and practices on his section and still be fortunate enough not to have an accident. On the other hand, if no dangerous conditions or practices are found by the Inspector and everything is satisfactory, the assistant foreman is given a credit of five merits; and each assistant foreman who does not have an accident during any month is given a further credit of five merits, which goes toward reducing the number of demerits standing against him until all the demerits are canceled, after which no further merits are given until he has again received demerits. The men are not permitted to accumulate merits, as this would have a tendency to make them more or less careless after they had accumulated a large number. The mine foreman is charged with all the demerits received by the various assistant foremen under him, and is credited with all the merits received by them. In the case of mine foremen, the premium for a month's clear accident record is $10 and the special premium for six continuous months' clear record is $15. Thus a mine foreman having a clear accident record of longer than six months will receive a bonus of $25 per month so long as his record remains clear, the assistant foremen under the same conditions receiving $15 per month. This premium or bonus is not considered as a part of the wages but is strictly in the nature of a reward for faithful services rendered to the company in the prevention of accidents. The mine foreman's account is not charged with any demerits of the assistant foremen when these are given for the neglect of duty or causes other than accidents. If a foreman or assistant foreman leaves the employ of the company and later re-enters it he assumes all demerits standing against him when he left.

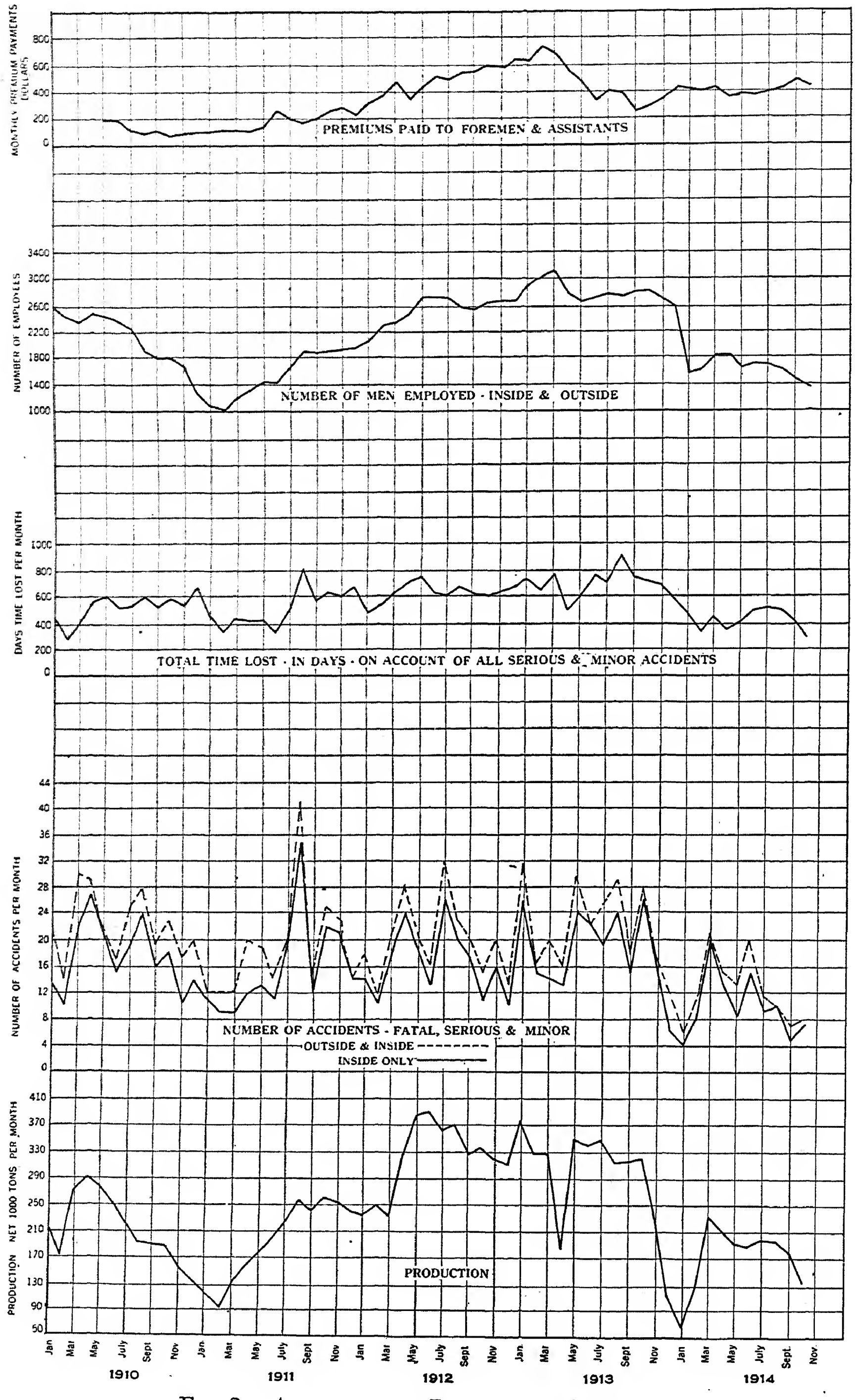


FIG. 2.—ACCIDENT AND PRODUCTION STATISTICS.

At the beginning of each month a statement is made showing the foreman and assistants at each mine, together with the number of demerits charged against them in the previous month, the number of demerits or merits received during the month, the number at the first of that month, the number of months' clear record, and the premium paid; and a copy of this statement is furnished each foreman and assistant foreman. This statement is looked forward to with a great deal of interest by the men and if any error is detected in it the matter is at once called to the attention of the person in charge of this work. The form of this statement is shown herewith.

United States Coal & Coke Company
Premium Statement for No Accidents—Month of September, 1914

Wks No	Name	Occupation	Demerits 9/1	Demerits Sept.	Merits Sept.	Demerits 9/30	No. mos. clean record	Premium	Spec. prem.	Remarks
2	T. J. McParland	M. For	....	10	20	....	8	10.00	15.00	
	Mike Cassett....	Asst.	....	.....	5	....	7	5.00	10.00	
	M. E. Maloney..	Asst.	....	.....	5	....	6	5.00	10.00	
	J. P. Hanley.....	Asst.	5	.....	5	....	4	5.00	.....	
	I. C. Yates......	Asst.	....	.....	5	....	15	5.00	10.00	
	Jas. Barrett.....	Asst.	20	10	....	30	.....	.....	.....	Not work full mo.

Since the adoption of the premium system in May, 1910, the company has paid 2,652 premiums, amounting to $19,005, or a cost of 0.15c. per ton.

In Fig. 2 a series of curves is given showing, since January, 1910, the production; the number of accidents, including serious and minor, both inside and outside, and inside only; the total time lost in days per month on account of serious and minor accidents; the number of men employed, both inside and outside, and the premiums paid to the foremen and their assistants for each month.

From Fig. 1 it will be noted that since 1909, when the accident prevention work was really started and actively followed, the number of tons produced per fatality, both inside and outside, has risen from 107,323 tons to 428,962 tons (to Nov. 1, 1914), and the production per fatal accident inside only has risen from 128,788 tons to 571,949 tons. The systematic timbering and the fact that all headings and places on which more than one car in a trip passes have ample clearances on *both* sides have been important factors in the reduction of accidents.

At each mine a Safety Committee of three workmen is appointed for a period of three months, and at least once in that time these men make a complete inspection of the mine and report any places which they find in dangerous condition or make any recommendations they can toward the safety of the mines. These reports are forwarded to the General Superintendent. The form used is shown herewith.

SAFETY THE FIRST CONSIDERATION

UNITED STATES COAL & COKE COMPANY

Report of SAFETY COMMITTEE No. ______ Works ______ 191__

NOTE: The Safety Committee will make their examination and recommendations beginning at the Coke cars on the coke yard and the cars on tipple sidings, through the tipple into the mine, including the coke yards, tipple, substation, haulage roads, traveling ways, hauling of the coal, mining, timbering, etc.; also including all machinery to see that it is properly guarded, or any other place in or about the works where men are working or machinery is located, including construction work.

REPORT OF CONDITIONS	RECOMMENDATIONS
(Do not crowd work. Use several sheets if required)	

SAFETY COMMITTEE

Name	Occupation
Name	Occupation
Name	Occupation

Noted:

______ Assistant Foreman

______ Foreman

______ Superintendent

NOTE: This Report must be sent to the office of the General Superintendent at the first of each quarter of each year.

Each month the General Safety Committee, composed of the General Superintendent, Chief Engineer and Mine Inspector, meets and considers all fatal or serious accidents which have happened during the preceding month, or any other accidents having any unusual features, and have before them, if possible, the victim or any witnesses to the accident, the assistant foreman in charge of the section on which the accident occurred, the mine foreman and the Superintendent. The statements of these witnesses are heard, their evidence is weighed, and measures to prevent similar accidents are recommended. In cases of neglect on the part of the assistant foreman recommendations are made as to punishment. A stenographic report of these investigations is made for our files.

Shortly after the first of each year, beginning in 1910, a meeting has been held, at which are present the general officials of the company and all of the superintendents, mine foremen and their assistants, and all outside foremen; the purpose of the meeting being to bring the assistant foremen into closer personal contact with the general officials than can be done in the ordinary course of business. After the inner man has been provided for, speeches are made by various members detailed for that purpose, about the best methods of preventing accidents or similar subjects, and ex-members of the staff, State mine officials or other persons interested in the prevention of accidents are present and give short discussions on subjects of interest. These meetings have undoubtedly been of great service in stimulating and maintaining the interest in safety work, and we believe they do more to impress the assistant foremen with earnestness in this work than any other single thing.

In the case of any accident, a letter is prepared in the General Office, giving, in a short manner, the details of the accident and impressing upon the foremen and assistant foremen the lesson to be learned from this and the necessity of guarding against similar occurrences. A copy of this letter is sent to each mine and posted on the bulletin board near the pit mouth. In case of serious or fatal accidents, the Mining Engineer visits the place and, where necessary, a sketch is made showing the details of the working place, and prints of this sketch are also sent to each mine and posted on the bulletin boards. In case of fatal accidents, inspection is also made by the company's Mine Inspector, Chief Engineer, or General Superintendent.

At the beginning of each month a print is sent to each mine, on which is shown the number of fatal, serious, and minor accidents which have happened at each mine during the current year and the number that have happened during the month just preceding. This print is posted on the bulletin boards, informing every one at the mine of the number of accidents which have happened at all the mines and showing the relative standing of each one in this respect. Such a print is shown in Fig. 3.

At various times, when an accident of unusual nature or unusual in-

SAFETY THE FIRST CONSIDERATION.
UNITED STATES COAL & COKE CO.
RECORD OF INJURIES FROM JAN. 1, 1915, TO FEB. 28, '15. INCL. SHOWN IN BLACK ●
INJURIES DURING MAY 1915 SHOWN IN RED ○

Mine	No Time Lost Inside	No Time Lost Out	Minor Inside	Minor Out	Serious Inside	Serious Out	Fatal Inside	Fatal Out	Date of Last Fatal Injury	Date of Last Serious Injury	Production Since Last Fatal Injury	Production Since Last Serious Injury
NO. 2 MINE	● ○								Feb. 6, 14 8	Oct. 10, 13 2	230,918 7	230,818 2
NO. 3 MINE	●●								Sept. 7, 12 3	Aug. 24, 14 7	313,426 4	34,298 10
NO. 4 MINE			●●● ○○○						Dec. 3, 13 6	Sept. 10, 14 8	161,246 9	47,311 9
NO. 5 MINE									Dec. 10, 14 9	Aug. 11, 14 6	20,620 11	57,312 8
NO. 6 MINE	●●● ○○								Aug. 24, 12 2	Nov. 14, 13 3	1,094,689 1	387,470 1
NO. 7 MINE			● ○			● ○			Nov. 25, 13 5	Mar. 1, 15 9	259,819 5	18,602 11

Mine	No Time Lost Inside	No Time Lost Out	Minor Inside	Minor Out	Serious Inside	Serious Out	Fatal Inside	Fatal Out	Date of Last Fatal Injury	Date of Last Serious Injury	Production Since Last Fatal Injury	Production Since Last Serious Injury
NO. 8 MINE									Nov. 21, 12 NOT WORKING	Aug. 2, 13	[illegible] 6	82,755 7
NO. 9 MINE			●● ○○						May 15, 13 4	July 28, 14 5	[illegible] 3	148,585 5
NO. 10 MINE									Aug. 19, 11 1	Sept. 19, 13 1	[illegible] 2	210,330 4
NO. 11 MINE			● ○						Jan. 17, 14 7	May 12, 14 4	140,467 10	99,953 6
NO. 12 MINE									June 14, 13 NOT WORKING	April 22, 13 4	167,332 8	225,072 3
POWER PLANT		● ○								July 22, 13		3,511,724
SHOP		●										

FIG. 3.—FORM OF ACCIDENT RECORD.

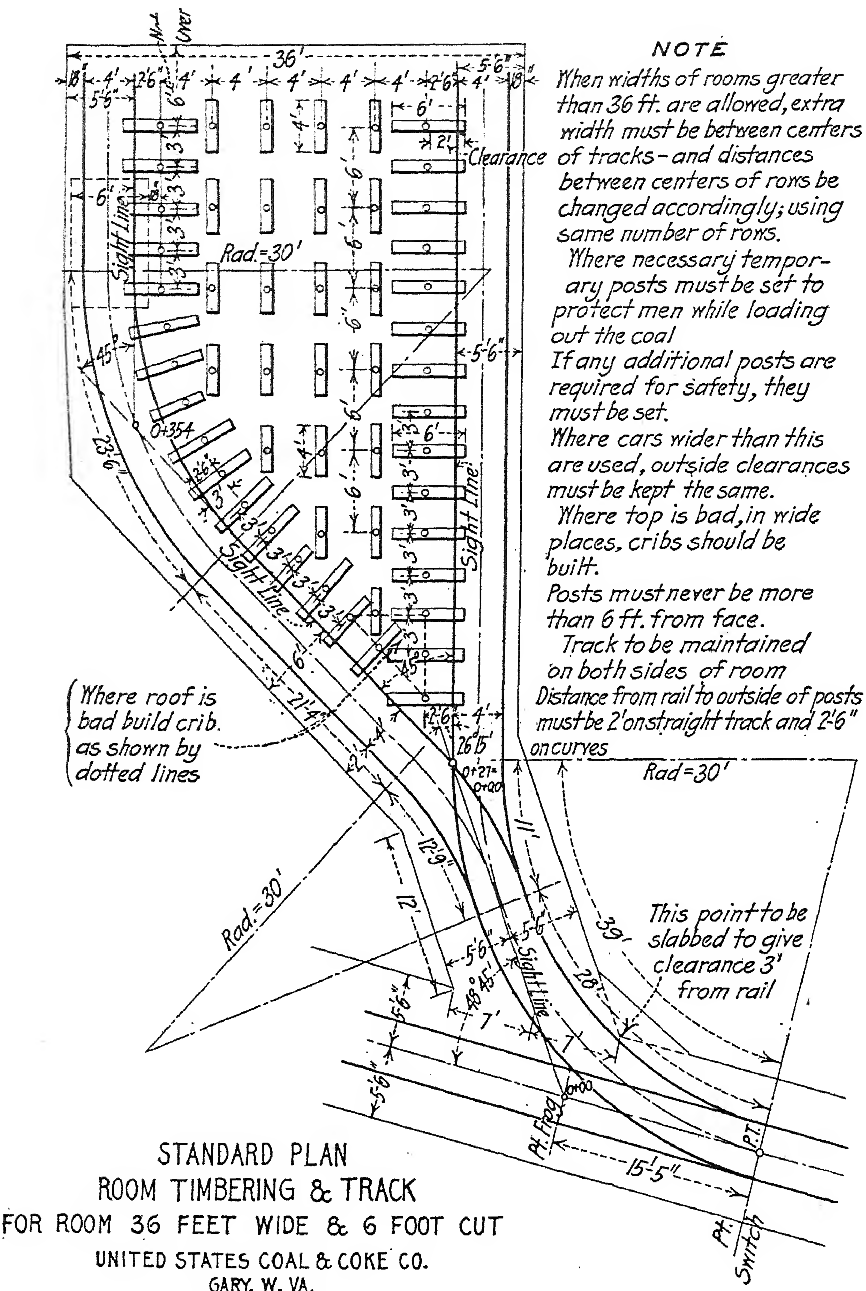
terest has occurred, a circular letter is sent to the assistant foremen to be posted in their sections, calling their attention to the matter and warning them against similar occurrences. In addition to this, safety signs, many of which are illuminated, are placed in the mines and around the plants, and are changed from time to time in order to stimulate interest.

Meetings of the assistant foremen, mine foreman, and Superintendent are held each week at each mine, at which the general details of their work are discussed, and particularly any accidents which have happened, and the steps which should be taken in their particular places to prevent similar occurrences.

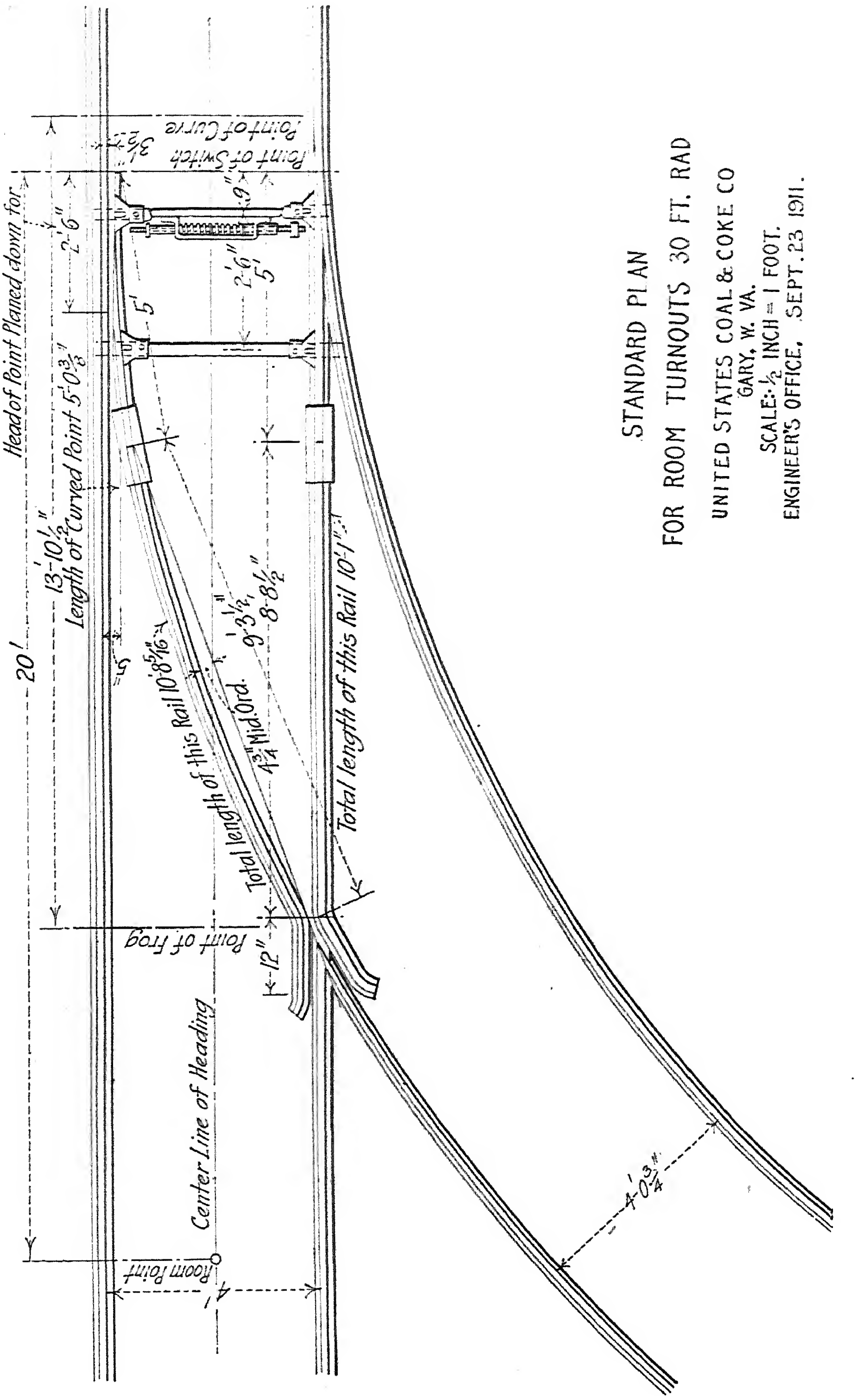
Last year, in co-operation with the United States Bureau of Mines, about 4,000 ft. of moving-picture film were taken, showing the complete details of the proper methods of mining. The object of these pictures was to show the miners exactly how the work should be done with safety and efficiency. These pictures have been exhibited at each of our different works a number of times; and it is our intention to repeat the exhibition from time to time in order that any new men coming in may be thoroughly posted. The class of men engaged in most mines, and, in fact, almost all men, can learn a great deal more through the eye by means of pictures of this sort than they can through printed or written instructions; and this film has been of great benefit in improving the efficiency and safety of our methods. Anything that tends to promote the efficiency of mining will undoubtedly, in the long run, promote its safety; and the safest mining will undoubtedly prove to be the most efficient.

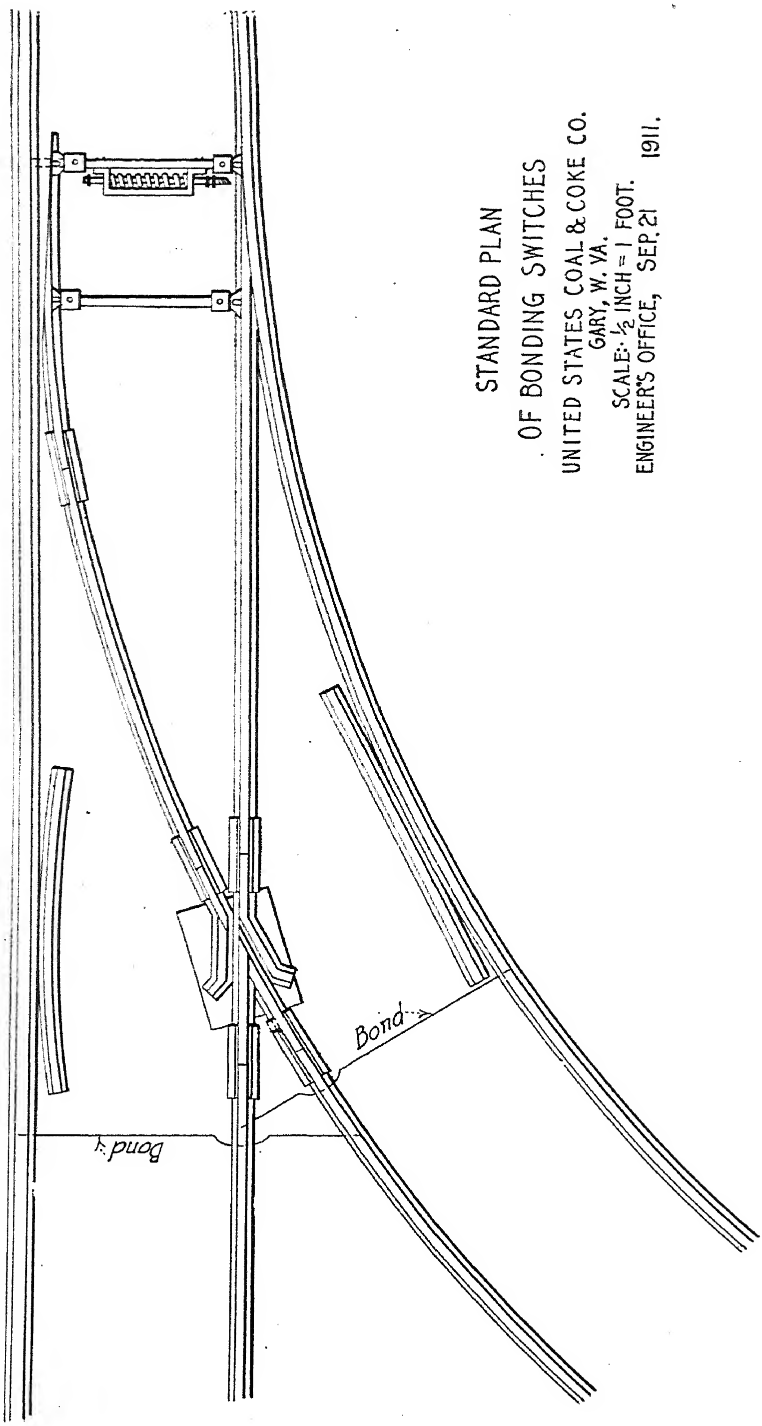
Accident prevention is, as has been said, very largely a question of education; and our experience has been that the great majority of men are willing to take the precautions their experience, or knowledge, has taught them to be necessary, and that they appreciate the efforts being made by their employers to prevent accidents of all kinds. The great question is to reach the man who is actually doing the work at the working face with the superior knowledge and experience of the higher officials, and this can only be done by a system of detailed supervision and instruction through bosses having the necessary time to see that dangerous conditions are removed promptly. When the man at the face learns to carry out instinctively the directions given in the rules formulated for his benefit, the accident-rate will not exceed one-fourth of the present one.

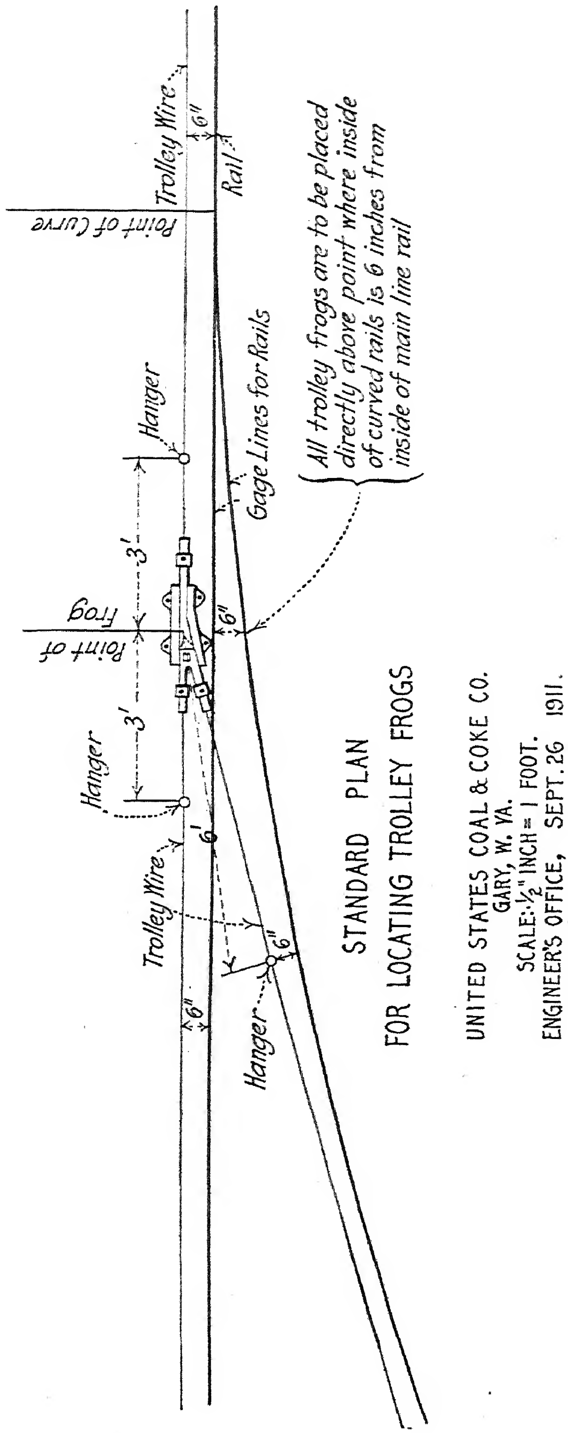
Any work of this kind requires unceasing vigilance, careful supervision, and the never-ending stimulation of the spirit of care in all those having anything to do with it.



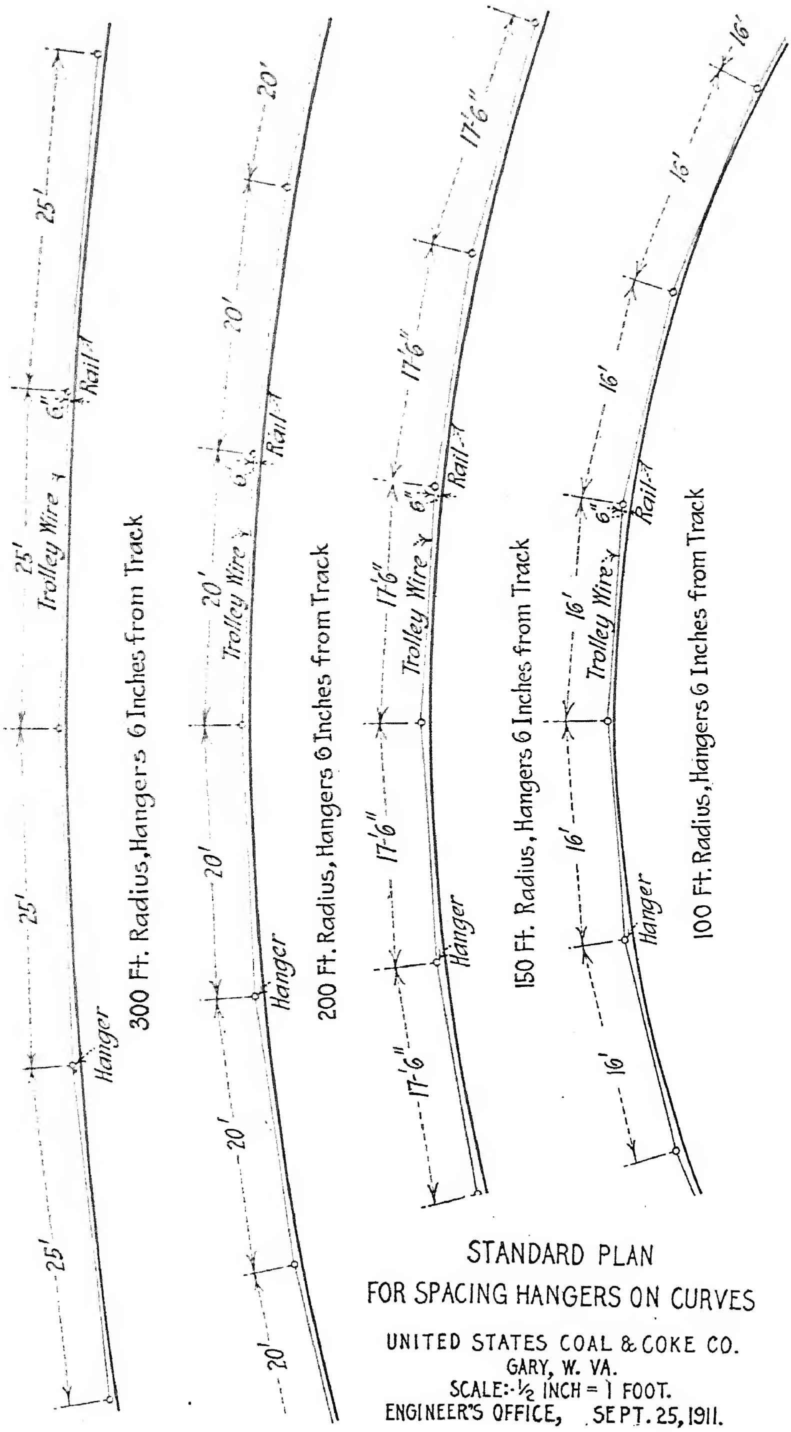
STANDARD PLAN
ROOM TIMBERING & TRACK
FOR ROOM 36 FEET WIDE & 6 FOOT CUT
UNITED STATES COAL & COKE CO.
GARY, W. VA.
SCALE 1/8 INCH = 1 FOOT,
ENGINEER'S OFFICE, AUG. 29, 1914
SAFETY THE FIRST CONSIDERATION


STANDARD PLAN
FOR ROOM TURNOUTS 30 FT. RAD
UNITED STATES COAL & COKE CO
GARY, W. VA.
SCALE: ½ INCH = 1 FOOT.
ENGINEER'S OFFICE, SEPT. 23 1911.
Head of Point Planed down for
Length of Curved Point 5'-0⅜"
13'-10½"
20'
2'-6"
3½"
Point of Curve
Point of Switch
9"
2'-6"
5'
5'
Total length of this Rail 10'-8 5/16"
Total length of this Rail 10'-1"
4¾" Mid. Ord.
9'-3½"
1'-1"
8'-8½"
Point of Frog
12"
Center Line of Heading
Room Point
4'
4'-0¾"


STANDARD PLAN
OF BONDING SWITCHES
UNITED STATES COAL & COKE CO.
GARY, W. VA.
SCALE: ½ INCH = 1 FOOT.
ENGINEER'S OFFICE, SEP. 21 1911.
Bond
Bond

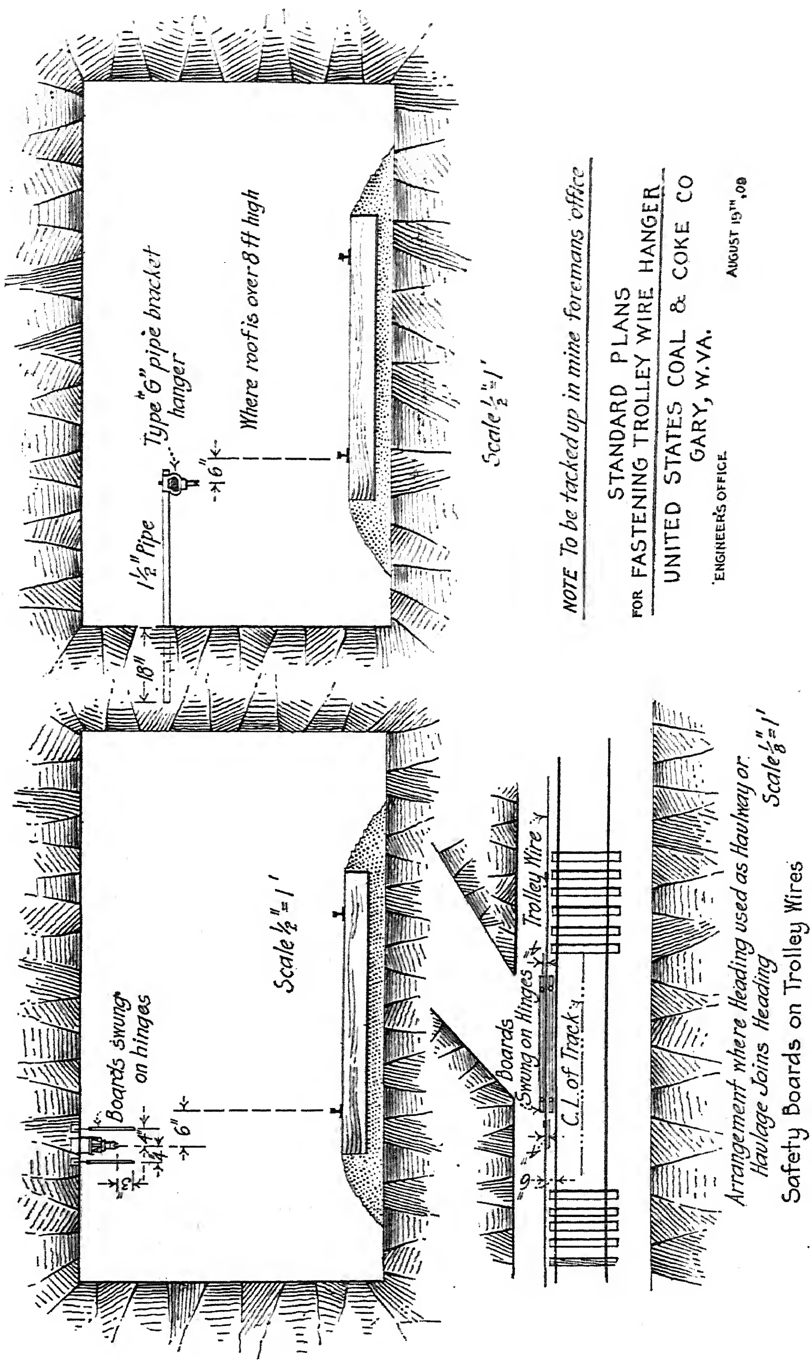


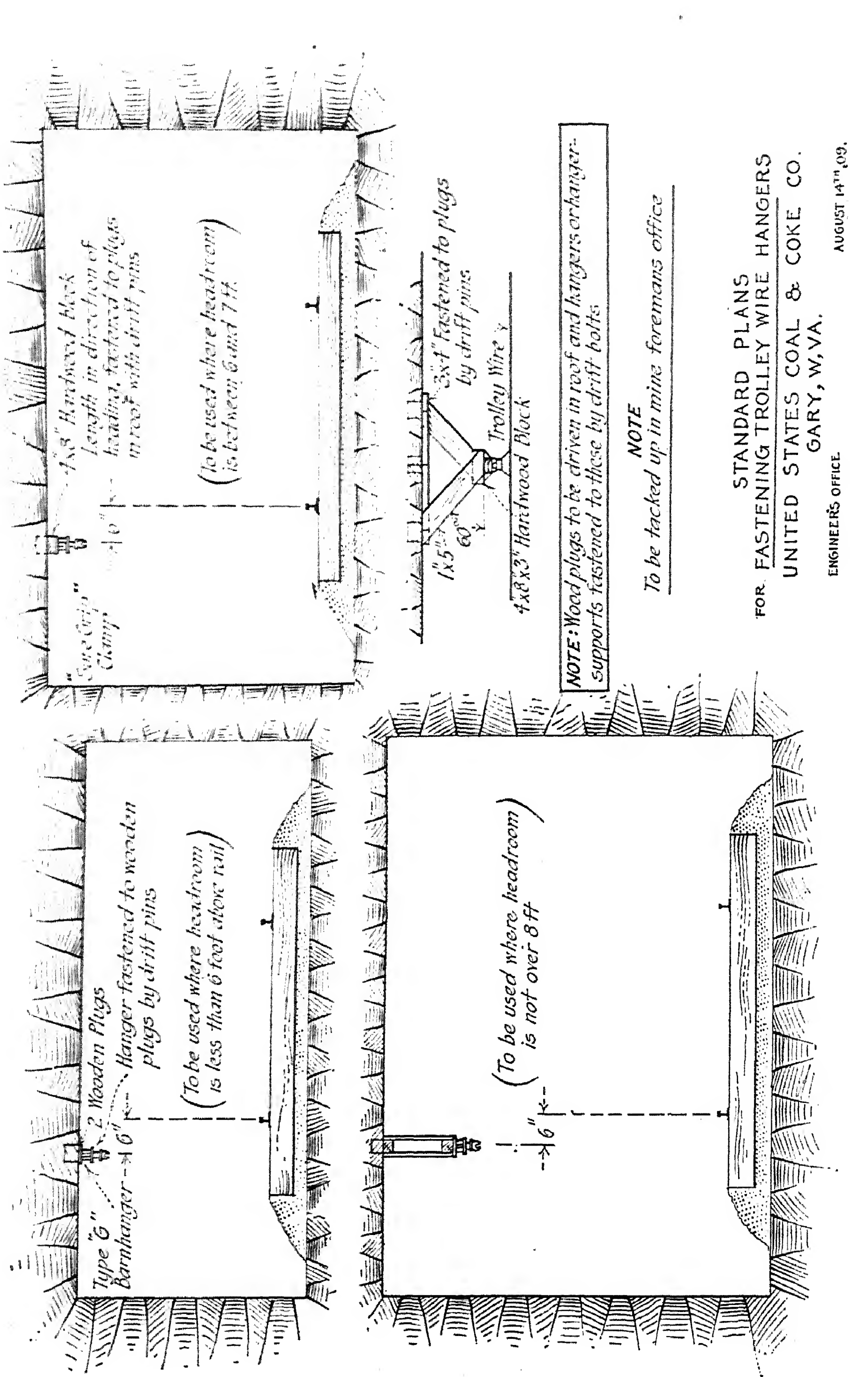
Trolley Wire
6"
Rail
Point of Curve
Gage Lines for Rails
All trolley frogs are to be placed directly above point where inside of curved rails is 6 inches from inside of main line rail
Hanger
3'
Point of Frog
6"
3'
Hanger
6'
Trolley Wire
6"
Hanger
6"
STANDARD PLAN
FOR LOCATING TROLLEY FROGS
UNITED STATES COAL & COKE CO.
GARY, W. VA.
SCALE: ½" INCH = 1 FOOT.
ENGINEER'S OFFICE, SEPT. 26 1911.

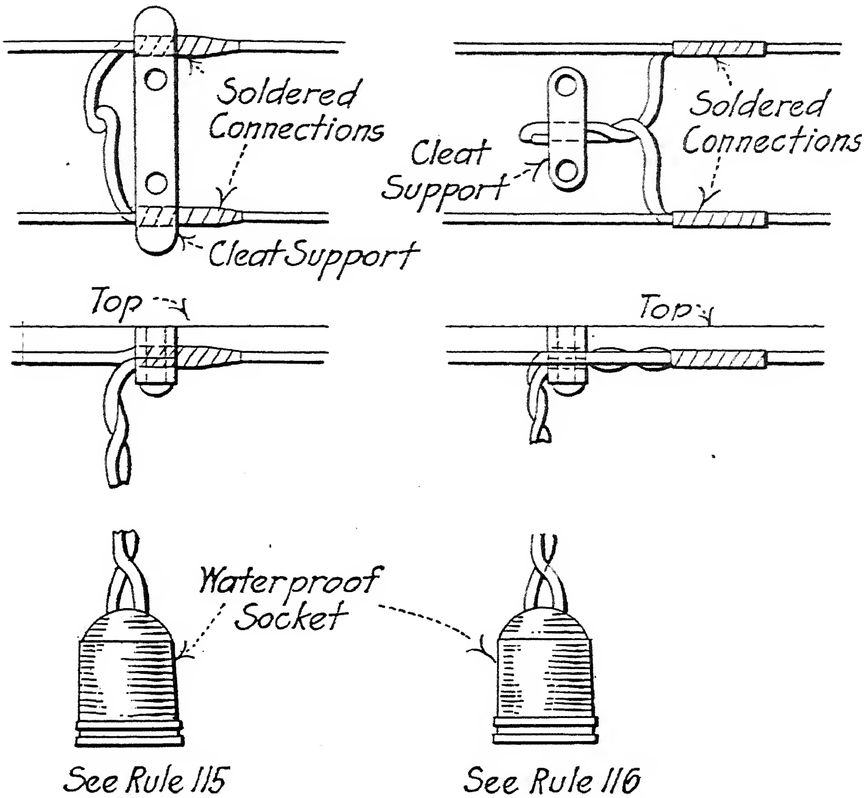


STANDARD PLAN
FOR SPACING HANGERS ON CURVES

UNITED STATES COAL & COKE CO.
GARY, W. VA.
SCALE:- ½ INCH = 1 FOOT.
ENGINEER'S OFFICE, SEPT. 25, 1911.


Type "G" pipe bracket hanger
1½" Pipe
6"
18"
Where roof is over 8 ft high
Scale ½" = 1'
NOTE To be tacked up in mine foremans office
STANDARD PLANS
FOR FASTENING TROLLEY WIRE HANGER
UNITED STATES COAL & COKE CO
GARY, W. VA.
ENGINEER'S OFFICE.
AUGUST 19TH, 08
Boards swung on hinges
3"
4"
4"
6"
Scale ½" = 1'
Boards Swung on Hinges
Trolley Wire
C. L. of Track
6"
Arrangement where Heading used as Haulway or Haulage Joins Heading
Scale ⅛" = 1'
Safety Boards on Trolley Wires


4x8" Hardwood Block length in direction of heading, fastened to plugs in roof with drift pins
6"
(To be used where headroom is between 6 and 7 ft.
3x4" Fastened to plugs by drift pins
1"x5"
60°
Trolley Wire
4"x8"x3" Hardwood Block
NOTE: Wood plugs to be driven in roof and hangers or hanger-supports fastened to these by drift bolts
NOTE
To be tacked up in mine foremans office
STANDARD PLANS
FOR FASTENING TROLLEY WIRE HANGERS
UNITED STATES COAL & COKE CO.
GARY, W, VA.
ENGINEER'S OFFICE
AUGUST 14TH, 09.
Type "G" Barnhanger
2 Wooden Plugs
6"
Hanger fastened to wooden plugs by drift pins
(To be used where headroom is less than 6 feet above rail)
6"
(To be used where headroom is not over 8 ft)



STANDARD PLAN FOR DROP LIGHT
UNITED STATES. COAL & COKE CO.
GARY. W. VA.
SCALE:- INCH= FOOT.
ENGINEER'S OFFICE, FEB. 23-1912.

Appendix A

SAFETY THE FIRST CONSIDERATION
UNITED STATES COAL & COKE CO.

SYNOPSIS OF INSTRUCTIONS ISSUED IN CIRCULAR LETTERS FOR MINE FOREMEN AND ASSISTANTS

Note.—These instructions are intended to supplement the mine law, and the posted rules of the company. Should anything in them conflict with the mine law or the rules, the law and the rules must govern and be followed explicitly.

In All Work Consider: First, Safety. Second, Quality. Third, Cost.

Approved, Sept. 25, 1911.

E. O'Toole, *General Superintendent.*

Safety The First Consideration

United States Coal & Coke Company

Standards for Ventilation and Inspection

1. The last breakthru of every pair of headings, whether working or not, must have at least 12,000 cu. ft. of air per minute passing thru it.

2. On room headings, the air current must be properly checked with canvas curtains, to cause it to circulate to faces of rooms. Curtains must not be more than seven (7) rooms apart.

3. Ass't. foremen must examine each working place in their districts, and mark each place visited with date of month, before allowing men to enter them. Each idle place must also be examined each day, and a record of the examination must be made daily in the book provided for that purpose.

4. Before entering the mine, ass't. foreman must ascertain the condition of the ventilating apparatus from the fan man or the substation man.

5. When the fan has been stopped for one hour, or more, for any reason, all places must be thoroughly examined by ass't. foremen before allowing men to enter them, and a record of the examination must be made in the record book.

6. All permanent brattices must be built of incombustible material—concrete preferred.

7. Temporary brattices must be made of boards, well put together and with joints stopped up with cement.

8. If it is necessary to carry a brattice from the last breakthru to the face of any working place, use boards and not canvas.

9. Air measurements must be made weekly and be recorded in the book provided for that purpose.

10. For each shortage of air reported by monthly measurement of engineers, ten (10) demerits will be charged to the ass't. foreman in whose district it was found.

11. Ass't. foremen must carry Pieler testing lamps when making examinations, and all final tests for gas must be made with this lamp. When an ordinary safety lamp will not detect gas, try the Pieler lamp.

12. The presence of dust, as well as gas, must be noted in the record book.

13. Dust must not be allowed to accumulate in working places nor anywhere in the mine.

14. All dusty places on haulage roads must be sprinkled daily.

Safety The First Consideration

United States Coal & Coke Company

Standards for Explosives and Shooting

31. No charge, of any explosive, in any one hole, shall exceed two (2) pounds.

32. Machine cuttings must be loaded in car before coal is shot down.

33. Before firing a shot, ass't. foreman must see that the place is free from gas and dust, properly posted and safe in all respects.

34. No one but ass't. foremen or persons designated by mine foreman are allowed to have possession of batteries or to fire any shot.

35. Before firing a shot, ass't. foreman must see that all workmen have withdrawn to a place of safety, and he must not be in the line of the shot himself.

36. Ass't. foremen must not fire any shot that is improperly drilled, drilled on the solid, or improperly tamped.

37. Ass't. foreman must not fire any shot within three hours after detecting gas in any place in his district.

38. Where more than one shot is to be fired in any place, one shot only shall be charged, tamped and fired, and an examination of the place must be made before firing the second or following shots. Only one shot shall be fired at a time. Ass't. foremen must carefully examine working places, after shooting, before allowing work to begin.

39. No explosives excepting flameless explosives, approved by the U. S. Bureau of Mines, shall be used.

40. All holes must be tamped with clay the full depth of the hole.

41. No shot that has missed fire shall be drilled out. Drill a new hole in such a manner that it will not come in contact with the explosive in original hole. Second hole must be charged lightly.

42. If shots miss fire, or are not fired for any reason, ass't. foremen must report the fact to the mine foreman, who must record the fact and the reason for it in his record book.

43. If necessary, water dust at working face before shooting.

44. Augers are to be made to gage, which must not exceed one and three-fourths (1¾) inches in diameter.

45. Use wooden sticks for tamping holes.

46. Shots must be fired by battery and not from machine or trolley wire.

47. Ass't. foremen must see that all caps or exploders are kept separate from explosives and at least thirty (30) ft. from them. Caps must be kept in a hole dug in solid coal.

Safety The First Consideration

United States Coal & Coke Company

Standards for Trackwork, Line Sights, Etc.

61. Line sights are to be carried five and one-half (5½) feet from the rib in rooms and air courses, and in the centers of headings.

62. Sights in all places working must be extended each day.

63. Sight lines must be marked on roof by continuous chalk lines in all places.

64. All haulage roads on which more than one car is hauled per trip shall be at least five (5) feet high above the rail, and there shall be at least two and one-half (2½) feet clearance between any part of a car and the side of the heading at all places.

65. Permanent track must be kept within one hundred fifty (150) feet of face.

66. Track must be laid to line.

67. Where grades are given, track must be laid to grade.
68. All rash, slate, etc., is to be kept cleaned off haulage roads.
69. Ties are not to be over eighteen (18) inches centers on haulage roads.
70. Loaders must lay the track in their working places.
71. See that steel ties are used for room tracks and that all straight track is laid to exact gage, forty-eight (48) inches.
72. Permanent track must be bonded as it is laid.

Safety The First Consideration

United States Coal & Coke Company

Standards for Timbering, Slate Work, Etc.

81. Where any dangerous slate is found, ass't. foreman must take it down at once before work is done near it, or anyone, or any trip, allowed to pass under it.
82. Loaders must post their working places. Proper arrangement and spacing of caps, posts, etc., is shown on standard plan attached, which must be strictly followed. Posts must be set in straight lines and be vertical. Cap pieces must be wedged tightly against roof, with wedge between post and cap.
83. Ass't. foremen must not O.K. nor allow any place to be cut where posts are more than six (6) feet from the face at the bottom.
84. All slate and loose rash must be taken down in haulage roads. Places must not be cut until slate or rash is taken down to face.
85. No permanent timbering will be allowed on haulage roads unless under special instructions from General Superintendent.
86. Slate or loose rash in manways is to be taken down, same as in haulage ways.
87. Airways are to be timbered in same manner as rooms.
88. Loaders must clean up slate falls in their working places, and must be paid extra for this.
89. Mining machines should start to cut on track side of place.

Safety The First Consideration

United States Coal & Coke Company

Standards for Electric Work

101. The general arrangement of hangers, etc., for trolley wire is shown on accompanying plans, which must be strictly followed.
102. Trolley wire must be hung six (6) inches outside of rail and must be as nearly parallel to it, both horizontally and vertically, as it can be.
103. Where roof is high, trolley hangers can be spaced thirty-three (33) feet apart on straight track. Where clearance between roof and wire is low, hangers can be spaced twenty-five (25) feet to twenty (20) feet apart on straight track; the wire must always be so hung that it cannot be forced against the roof by the trolley wheel passing under it. On curves, hangers are to be spaced as shown on standard plan, depending upon the radius of curve. Hangers must be close enough to prevent being bent sidewise by strain of wire. Trolley wire must be pulled tight and be dead ended with an insulated turnbuckle to an anchor bolt in roof or to a heavy post.
104. Where trolley wires or feed lines cross a heading or at turn out for side tracks, each must be protected by hanging boards, so that neither man nor mule can come in contact with wire.

105. Section line switches and sectional insulators must be installed on all trolley wire branches. Section line switches must be installed on machine and pump lines. Cable for wiring switches and insulators must be of same capacity as circuit controlled by switch. Ends of cables must be soldered solid and filed to fit terminal, or be soldered solid into feeder ear or terminal of switches and sectional insulators. Cables must be kept clear of switch boxes and porcelain tubes be used to bush holes, or holes have one-half (½) inch clearance all around cables.

106. Trolley frogs must be used at all turn outs and each frog must have an electric lamp beside it. Frogs must be located and supported by extra hangers, as shown on standard plan.

107. A clear space of at least two (2) inches must be allowed above trolley wire where it passes through a door, and hangers must be placed close to each side of door to prevent wire being forced up against wood.

108. Joints on trolley wire must be made with sleeves.

109. Feeder cables must be supported at intervals of twenty (20) feet on barn hangers and special "Gem" insulators. Cables are to be placed twelve (12) inches outside of trolley lines and must be connected to them at intervals of two hundred (200) feet by feeder cars and solderless cable taps. Cables must be properly dead ended with cable clamps and insulated turnbuckles.

110. Machine wire must be supported on insulators, which must not be more than thirty-three (33) feet apart, and close enough to prevent wires touching posts, rib or roof. Ground wires must be put up in same manner as live wires. Where roof is good and not less than six and one-half (6½) feet above top of rail, wires should be put on roof at side of heading, and be eighteen (18) inches apart, the live wire being placed as close to rib as possible; where necessary to place wires on ribs and posts they must be twenty-four (24) inches apart. All wires must be dead ended on porcelain insulators. Porcelain tubes must be used around wires passing thru doors or curtains, and insulators must be placed close to each end of tubes. Joints must be well twisted together, with six (6) turns on each side. No hook joints will be allowed. Machine wires must be placed on same side of track as trolley wire. All wires must be pulled tight and be well tied to approved insulators. Machine wire should not be used in rooms less than four hundred (400) feet long. Where rooms are longer than this, room wires must not be connected to heading wires, excepting when machine is in operation in room. Each machine must carry one pair of jumpers to reach from heading to room wires. Insulators must be put on pins, and no wedging of insulators between wires or between wire and roof or coal or wood will be permitted. One wire only must be on one insulator.

111. All tracks on headings where electric haulage is used must be bonded when laid. Holes for bonds must be clean and bright and bond terminals be forced firmly against the metal. Splice bars must be left off joints until bonding is inspected by mine foreman or superintendent. All switches on haulage roads must be bonded as shown on standard plan attached, and all track should be cross bonded at intervals not exceeding five hundred (500) feet.

112. All light wiring must be thoroughly insulated from roof, rib or timbers, and must be tied to porcelain insulators with non-conducting material. Wire from lamp to ground must be on insulators, and wire from rib to rail be buried. No nails or staples will be allowed for fastening wire.

113. All insulated cables and wires must be kept free from grounds, same as bare wires.

114. Pipe lines must not be used solely for pump motor returns, but when running close to and parallel with a bonded rail, may be connected to rail at intervals not exceeding five hundred (500) feet.

115. Where lights are to be installed in mines, or damp, wet or dusty buildings,

where waterproof sockets must be used and where lamp is not to be carried about, wires to socket must be soldered direct to the circuit wires, but supported independently of them. Porcelain cleats or split knobs can be used to support droplight. See plan.

116. Where lights are to be installed that are used for portable purposes, or where lights come in contact with surrounding objects, portable cord—*not lamp cord*—must be used, wires to be soldered direct to circuit wires and be supported independently of them with porcelain cleats. See plan.

117. Lamp cord will not be approved in wet or dusty places. Cord must hang free from rosette. No looping of long cords and no suspending from hooks or nails in ceilings will be permitted.

Safety The First Consideration

United States Coal & Coke Company

General Rules

121. Mines will begin work at 7 A. M. and stop dumping at 3:45 P. M.

122. All men must be checked in and out of the mine each day.

123. No one but employees are allowed to enter the mines unless accompanied by the mine foreman or some other official.

124. No person under eighteen (18) years of age shall be employed without first furnishing affidavit of age from his parents or guardian.

125. All work, excepting such repairs as cannot be done while operating or at night, must be suspended on Sundays.

126. Do not allow grease, oily waste or any other inflammable material to accumulate in pump houses, offices or anywhere else.

127. Do not allow wood, old boards, ties, posts or any inflammable material to accumulate in the mines.

128. Mining machine men must be in working places or at mine office at 4 P. M. to receive instructions from ass't foremen.

129. Unless otherwise instructed, one man only will be allowed per working place.

130. No machinery of any kind must be allowed to operate unless all gears and dangerous portions are fully guarded.

Discussion

William H. Grady, Bluefield, W. Va.—Mr. Eavenson gave a comparison between accidents in the State of West Virginia and in the Pocahontas field. The figures quoted are, I presume, taken from the reports of the State Mine Inspector. There are no statistics available, in readily accessible form, giving a comparison between the Pocahontas field and the mines of the United States Coal & Coke Co. which are a part of the field.

It occurs to me that the paper would be more complete with these data and I will say that the mines of the Pocahontas field produced during the year 1913 about 15,000,000 tons, about 3,000,000 tons of which was produced by the United States Coal & Coke Co. This gives an indication of the extent of its operations.

The Pocahontas field lies largely in the counties of McDowell, Mercer,

and Wyoming in West Virginia, and Tazewell in Virginia. Without consulting maps, I should say that all of the acreage under operation at present by this company lies wholly in McDowell County and that the average conditions of roof top and bottom in its mines are the average of the county, but that the worst conditions in some of these mines are the worst to be found in the county. The average conditions in Tazewell County are about the same as in McDowell County, except that the average thickness of the seam is much greater. The average conditions of top and roof and bottom are better in Mercer and Wyoming counties than in the mines of the United States Coal & Coke Co.; the worst conditions in these counties are not so bad as in McDowell and Tazewell counties and the best are better than those encountered in McDowell and Tazewell counties; also the thickness of seam is less in Mercer and Wyoming counties.

THOMAS W. DAWSON, Scottdale, Pa. (communication to the Secretary*).—The H. C. Frick Coke Co., the mines of which are in Fayette and Westmoreland counties, Pennsylvania, in the Connellsville coke region, has been pursuing a safety campaign for a number of years and a short *résumé* of its practices will be of interest in connection with Mr. Eavenson's paper.

Every official and foreman of the company is continually impressed with the fact that "safety" should be the first consideration, and all officials and their subordinates are brought together as one great committee on safety. Pamphlets showing the duties of the miner and the manner in which he may protect himself from danger and giving safety regulations for those working around machinery have been printed and generally distributed. Permanent danger signs are placed wherever there is the least possibility of an accident. When men are working in shafts, the "Men in Shaft" sign is placed so that no accident can be caused by mistake in moving cages. A similar sign is placed on hoisting engines and other machinery when it is being repaired. When workmen are cleaning or making repairs to the inside of a boiler, a "Man in the Boiler" sign is displayed outside and the steam valve for this boiler is locked and the key carried by one of the men until the work is completed. When coke-drawing machines are being repaired or cleaned, the "Do Not Move" sign is placed on the controller and the trolley wheel is locked and the key carried by one of the repair men until the work is finished. "No Clearance" signs are conspicuously displayed at all points about the plants where there is no clearance for a man between moving cars and obstructions of any character. Bridge guards and overhead warning signs are placed wherever needed.

In the mines, guide signs in various languages are posted at road

* Received Jan. 9, 1915.

junctions and on traveling ways, indicating the safest way out of the mine.

All machinery is safely guarded. These guards include locking devices for handwheels of valves, safety locks for electric switches, guards for water gauges; safety gaskets to be inserted in steam blow-off and feed-water connections when cleaning and repairing boilers; safety locking device for self-dumping cage; soap lubrication for air compressors; wagon guard and dumping platform for swing-gate mine cars; spooling device for tail ropes on haulages; stiles or protected crossings over rope and sheaves where necessary for men to pass; improved safety catch for cages; device for positively rectifying wagon catches on car hauls; self-closing hinges for shaft gates; steel galleries for runways over boilers, and safety platforms for operating electric larries. "Do Not Touch" signs are used about electric wires, indicating voltage of current; and "Do Not Pass Under" signs are used where there is danger in passing underneath structures. Steel doors are provided to drop over shafts which have wooden head frames or coal bins above them, should these wooden structures catch fire. The company has originated a device for automatically controlling high-pressure air compressors. When the temperature of the discharge air in the pipe reaches a predetermined point, showing that the pressure is excessively high, it acts on the thermometer and recording device, thus closing an electric circuit and energizing a solenoid. This moves over a tripping device, which opens the pilot valve, releasing the steam pressure on one side of the regulating piston. Thereby the valve on the steam feed pipe is automatically closed, shutting off the steam and stopping the compressor.

All hoisting engines are equipped with an automatic overwinding device, which acts directly on the engine, cutting off the steam and applying the brakes.

When it is necessary to clean the sump at the bottom of the shaft, the cages are hoisted to a clearance height and secured by iron pins, through holes in the guides; these pins are attached to the guides by chains, which prevent their removal when not in use.

At the surface landings of all shaft mines, a device is installed which prevents the gates from being opened when the cage is not in position at the landing. All hoisting compartments of shafts are lined at the cage ends. All cages and safety catches are periodically inspected, tested, and a written report made of the inspection. In no case is a hoisting rope kept in service longer than 2½ years, even though apparently safe and in good condition. Frequent inspection of air shafts must be made to keep them open and free at all times from ice and other obstructions. A fireboss must make this examination and travel either up or down such shaft once each day, the mine foremen once each two weeks, and the superintendent once a month.

The company's rules require that in mines generating explosive gas not less than 500 cu. ft. of air per minute per person employed in the mine shall be provided at the intake and this must be so distributed that there will not be less than 300 cu. ft. per minute per person employed in each split at the working faces. No mine shall have at the intake less than 300 cu. ft. of air per minute per person employed, and at the working places at least 150 cu. ft. per minute per person employed. Measurements of air supplied are carefully made and reported to the general office once each week. Local officials at mines generating gas are required to keep air up to the working faces and to such other places where explosive gas might be encountered. At a number of the larger and more recent plants, the ventilating fan is operated by two engines, one on each end, and either of them powerful enough to operate the fan in case of failure of the other. All ventilating systems in the mines are ascensional.

The Clowes hydrogen-test lamp is used in all mines generating gas, for testing purposes. Samples of air are taken in gaseous mines and sent in copper cans to the company's laboratory, where they are analyzed. The results of the analyses are reported to the general office and to the mine. If these show a percentage of explosive gas which might have been detected by the Clowes lamp, the party making the test and reporting no gas is required to make an explanation.

Boreholes are frequently drilled from the surface to release any dangerous accumulations of explosive gas in the gob, where these cannot be removed by the mine ventilation. Shot firers have been employed to do all blasting by battery, and inspect all places where shots have been fired to see that there is no fire or other danger thereafter. Only the safest permissible explosives are used, and all tamping is done with clay.

All safety-lamp mines are examined on Sundays, holidays and lay-off days, and all mines which have been idle for more than two consecutive days are examined before operations are renewed. In the larger mines, wherever safety lamps are used, auxiliary escapeways are provided. In some instances these are stair shafts from the surface to the mine, placed in the active working sections, and used also for additional ventilation. In other cases, means of escape are provided by having connections between mines, which are closed by double iron doors. Frequent examinations are made to see that these doors are always in condition for use. Where coal dust occurs, a system of pipes and a supply of water under sufficient head and all necessary appliances are provided to dampen thoroughly the floor, sides, and roof of all parts of dry mines.

On rope haulage, a device is provided for disengaging the rope from the trip as soon as it is given slack. Brakes are provided for all mine cars and 2½ ft. clearance is provided on all haulageways on one side; this side being indicated by a wide whitewashed strip on the rib.

Systematic timbering systems are devised and strictly followed.

Printed regulations cover the system of timbering in rooms, headings, and rib and pillar drawing: these are worked out to suit conditions at the various mines. Timber is not set without caps or crossbars.

All mines have complete mine-telephone systems. Stables, pump rooms, haulage-engine rooms, shaft bottoms, underground offices and all such places where men might congregate are of fireproof construction and are kept clean and neat. No open lights are allowed in any building. Cans are provided for the reception of oily waste, grease, small quantities of oil, etc. All electric wiring is carefully inspected twice each year. All bare power lines underground and on the surface are properly guarded for their entire length by a neat wooden guard, so as to prevent the workman or his tools from coming in contact with the same. For the same reason, trolley wires for coke-drawing machines are placed at a sufficient height to make contact with tools unlikely. A system of checking men in and out of the workings is maintained at all of the mines. All abandoned places in the mines are fenced off.

The company employs four mine inspectors, one of them acting as chief. It is the duty of these men to visit each mine and thoroughly inspect it at least once every 60 days. When an accident occurs in or about any mine, the chief mine inspector promptly visits the scene of the accident, gathers all of the data he can relative thereto and makes a sketch of the surroundings. This sketch is put into permanent form, blueprinted and sent to each mine with a circular letter, giving a full account of the accident and making suggestions for the prevention of similar ones. This is discussed at the meeting of the local officials at each plant. Once each week, the superintendent of each plant and his subordinates meet and discuss mine conditions and operations in general and especially matters pertaining to the safety of their employees. The discussions of these meetings are reported to the General Superintendent each week. General meetings are held at stated intervals at the general office, which are attended by the superintendent of each plant and heads of departments.

Projections for mine workings are made far in advance of the actual work, and the haulage and ventilating problems are planned so that when the mine is developed the best system is in use. Specifications are written for each mine, stating where and how the mining is to be done. The officials of the company make detailed inspections at intervals, insuring that their instructions and the best methods are actually followed.

A safety committee of three or four men is appointed at each mine, which inspects periodically the working places, roadways, ventilation and any other things which in its opinion might be the cause of an accident. The committee reports in writing to the superintendent of the mine, who forwards the same to the general office. These suggestions are im-

mediately acted upon and all dangers reported, should there be any, are removed as quickly as possible. Three rescue and first-aid stations are maintained at the different plants of the company, which are fully equipped with the best apparatus and accessories obtainable. About 400 men have been thoroughly trained and qualified in both rescue and first-aid work, local contests being held by the different teams at various times.

Emergency hospitals, fully equipped, have been provided at a number of the largest mines and the company is contemplating the erection of one at each of its operating plants.

Tests are made frequently for gas above roof falls in gobs. Work is prohibited in any place in which gas is found, until after it has been removed. Mine inspectors instruct all new employees about dangers of their work. In the accompanying table is given the accident record of the H. C. Frick Coke Co. for the last five years.

Fatal-Accident Record of H. C. Frick Coke Co.

Year	Total Coal Produced, Tons	Fatal Accidents Inside of Mines	Fatal Accidents Outside of Coke Plants	Total	Based on Fatal Accidents Inside of Mines		
					Tons Mined per Accident	Accidents per Million Tons	Accidents per Thousand Employees
1910	16,567,609	30	10	40	552,253	1.81	3.20
1911	14,993,417	23	4	27	651,888	1.53	3.32
1912	18,596,502	33	7	40	563,530	1.77	2.91
1913	18,097,038	36	10	46	502,695	1.99	3.44
1914	11,725,448	12	4	16	977,121	1.02	1.46

Charles W. Goodale, Butte, Mont.—I had the pleasure of visiting Mr. Eavenson last October, when he showed me through the plants of the United States Coal & Coke Co., and explained the merit and demerit system by which such great results were obtained.

I have charge of the safety work of the Anaconda Copper Mining Co., at Anaconda, Mont., and was anxious to see if it would be possible to apply the merit and demerit system in the work of our company, but the trouble with us would be that under the conditions existing in our mines the foremen would not be able to take charge of the same number of men. In the mines of the United States Coal & Coke Co. it is possible to give each assistant foreman charge of about 25 men, but those who are engaged in the mining of copper will appreciate the difficulty in equalizing the number of men that the sub-foremen or assistant foremen can take charge of. In one case it might be easy for one assistant foreman to have charge of 40 men, working on one level, whereas in another mine in the same district it would be very difficult for a foreman to take charge

of more than 20 men, the foreman having to go from one level to another. I have held the belief that there was a way of working out a proportional award, or something of that kind, based on the merit system, and depending on the number of men employed under each foreman, but we have not succeeded yet in doing that.

I thought Mr. Eavenson's work would show an increased efficiency, and judged from some of the figures I was able to work out that it would do so, but for the year 1913 it does not seem to have worked that way, although I presume his answer will be that they were not mining so much coal that year, and the tons of coal per man would be reduced on that account. I would like to have him explain whether the close attention to the accident record did not bring about increased efficiency.

I submitted a copy of his paper to one of my friends in Montana, who is engaged in coal mining. He did not care to be quoted, but in regard to the matter of standard timbering, he says:

"If I were to criticize the paper, it would be on the plan of standard timbering. While this is customary in quartz mines, I do not think it is applicable to coal mines where the variation is so marked between good roof and poor roof in the same mine, and my fear would be that in conforming to a standard plan some of the extraordinary pieces of bad roof might be passed over or considered as having been taken care of by timbers within a few inches of same and yet not just where they ought to be. This is partly provided for by the provision that extra timbers are to be used where necessary, yet a timber close by might in cases be considered as having taken care of the condition."

In regard to the question of the effect of intoxication, the Anaconda Copper Mining Co. is now publishing a monthly paper called *The Anode*, so named because all the copper shipped from the Anaconda mines goes out in the form of anodes. In the February number is an item in which you may be interested. It appears as follows:

"*Intoxication*

"Among the rules appearing on the bulletin boards at the mines is the following:

"Never go to work after drinking liquor, and if you must drink, Stay Home. Experience has proven that a great many accidents are caused from drinking intoxicating liquors."

(Some of you may know of the fact that on Sept. 1, 1914, the State militia was sent into Butte to stop some disorder, and Butte was put under martial law temporarily.)

"It will be remembered that from Sept. 1, 1914, to Sept. 14, all saloons in Butte were closed; that from Sept. 14 to Sept. 24 they were open only

from 8 a.m. to 7 p.m., and that for the balance of the month they were open only from 7 a.m. to 10 p.m.

"The accident record of the Anaconda Copper Mining Co. shows the following significant figures:

"Number of accidents per 10,000 shifts:

July	6.22
August	11.25
September	4.21
October	7.58
November	6.07"

In other words, the number of accidents per 10,000 mine shifts dropped from 11.25 in August to 4.21 in September, and came up again in October to 7.58.

I presume it is the experience of everybody that the drinking of intoxicating liquors has a great deal to do with accidents. Men are not only liable to accidents themselves when drinking intoxicating liquors, but they are careless while under the influence of intoxicants, and this carelessness leads, perhaps, to the injury of their fellow-workmen.

I fully agree with Mr. Eavenson that education is a very important thing in the efforts to reduce the number of accidents. We found by a careful review of our accidents at Great Falls that 85 per cent. of them were due to the carelessness of the men themselves, or their fellow-workmen, and that not over 15 per cent., by close analysis, could have been prevented by any amount of safeguards or precautions introduced by the company. We are endeavoring to educate the men as far as possible to have them feel we are in earnest about the "safety first" movement. Recently we went through the mines and had a lot of photographs taken, also some moving-picture films, showing regular practices of the men, and special pictures showing bad practices. We intend to get the men together and have them see these pictures as soon as possible. I realized the importance of this method of education at our October meeting of the Institute in Pittsburgh, when we were shown some very good pictures, taken by the United States Bureau of Mines, of accidents occasioned by bad practices in coal mines. I do not believe they have succeeded in getting the same kind of pictures in metal mines. It is important to have our men see some pictures illustrating bad practices in our own experience, and I hope to have these pictures on exhibition at an early date.

J. Parke Channing, New York, N. Y.—I think this is the proper time to put on record a conversation which I had the other day with a prominent Russian banker who came to New York to represent his government in connection with certain financial negotiations. I was introduced to him by Chester W. Purington. What he said to me, which

was so interesting, was in reference to the abolition of the use of vodka in Russia. He said that he was interested in some coal mines in Russia, and that when the war broke out 60 per cent. of their men were drafted into the army and the production of the coal mines went all to pieces. Simultaneously with that, the use of vodka was prohibited. The manager was in despair: there was a great demand for coal, and with his lessened force he did not know what he was going to do. In about a week the production of coal commenced to pick up, and pretty soon to increase very materially, and the banker telegraphed to the manager to know whether he had secured any more men. The reply came back that he had not secured any more men, but was still working with the 40 per cent. of his regular force. The production gradually increased, and to make a very short story of it, at the time that he left Petrograd, the mine was then working with about 50 per cent. of its normal force, about 10 per cent. having been added to the 40 per cent. which was left when the draft was made on the men in the mine. The production, however, was 130 per cent. of the best production at any time before the war.

He then told me the gradual steps. During the time of vodka drinking the men worked about four days a week. When they were deprived of vodka, they wanted something to do, and as times looked as if they were going to be hard, the men worked seven days a week. After they had worked for a month or so on that basis, they voluntarily came to the manager and asked that they might be relieved of the day's pay method and be put on contract. This he readily agreed to and put them on a tonnage basis. The tonnage still further increased, and then the men found that it was impossible for them to work seven days a week. You will remember after the French revolution they tried to cut out Sundays, and to work every day of the week. In the Russian mines referred to the men themselves finally concluded they would only work six days a week, and so in their present condition this increased efficiency is achieved with six days' work a week. He said that the physical aspect of the men had marvelously improved, that their financial condition was better and they wore better clothes. He said the most remarkable thing was that some of the men 40 and 50 years old had started in to learn to read and write. So that outside of the question of safety, from the efficiency standpoint alone, you can see what the prohibition of alcohol produces.

The State of Arizona, I will admit much to my surprise, went dry the first of last January. On Jan. 17, in the Miami-Inspiration district, we had a strike in which the men were out eight days. It was the most orderly strike I ever heard of. Negotiations were going on during that time, and finally a sliding scale was arrived at, and eight days after the strike the men went to work. During the whole of that time there was not a single incident of disorder on the streets, during the day or at night.

My manager wrote me that there was not even any loud talking. That is another example of what the abolition of alcohol will do.

B. F. Tillson, Franklin Furnace, N. J.—Very successful results have been obtained by the New Jersey Zinc Co. at the Franklin mines due to the installation of a bonus system in the payment of the shift bosses or under foremen for the achievement they have made in the reduction of accidents. As a comparison, if we consider the first two months of the year 1913, no bonus system was then in vogue, although numerous safety measures of the general sort that most mines employ had been in use for a number of years. In May of that year a bonus system was instituted, whereby the shift boss under ground who had the best record for all the remainder of the year received a bonus of $200. This stimulated the shift bosses to their best efforts. The rating was worked out rather elaborately on the basis of the employers' compensation laws, making a stated value, within limits, for various types of accidents, whether minor injuries or serious injuries, and a demerit representing that value placed against the shift boss's record; the shift boss having the least amount of demerits, for the year, received the bonus. It was very pleasant for all of us to know that the man who actually did receive the bonus was the man whose work had appeared throughout the year to show the greatest interest in safety precautions, although his territory was probably one of the most dangerous in the mines.

At the beginning of the year 1914, it was realized that there were some weaknesses in this system, inasmuch as the likelihood of the miner receiving a slight injury was as great as that of a serious injury; in other words, that good luck played an important part; and whether a fall of the ground merely bruised his foot or crushed his skull was often a question of good fortune. The system was reorganized at that time and placed on a basis of the number of accidents entailing loss of time, measuring a loss-of-time accident as any which incapacitated a man from returning to work the day following that on which he was injured. Lost-time accidents were rated against the number of shifts of labor which were performed in any shift boss's gang during the month. The standard rating then adopted was that of 1.2 disabilities (accidents resulting in loss of time) per thousand shifts of labor which were worked in any boss's territory. This worked out very favorably for our safety work. As a comparison, taking the same basis of disability rating for the first four months, before any bonus system had been installed, approximately 25 per cent. of the shift bosses working in the mines would have received a bonus. During the eight months in which the lump-sum bonus was paid to the man having the best record for the year, the standing would have been that about 40 per cent. of those who could possibly have received bonuses would have received bonuses under that rate.

During the first nine months of the year 1914, 58 per cent. of the possible recipients of bonuses received them for having a disability rating of less than 1.2 per thousand shifts of labor worked. Of that 58 per cent. 36 per cent. of them had an absolutely clean slate, no disabilities in those months. In other words, 36 per cent. of the number of shift-boss months which were possible showed no disabilities in the working force from accident.

Mr. Goodale brings forth the feature of the application of this bonus system to metal mining, where we have territories at different levels, and the shift boss often has a large area to cover, and on different levels. The sizes of the gangs which were rated under this ruling average from about 30 men to as many as, in some cases, 75 men. Naturally, it seemed that the man who had the largest number of men under him incurred the greatest risk. The saving feature was the pro-rating according to the number of shifts worked, so that the man who had 75 men working in his gang could afford to have one or two disabilities per month and still be a participant in the bonus, whereas the man who had only 30 men working for him had to have a clean slate to participate in the bonus.

The question of the payments of bonuses to the bosses is one which has been more or less argued by those interested in safety work, and it seems to me it is of vital interest that it has worked out so well in this particular trial.

Mr. Goodale raises the question of the accidents based on rate of production, which seems to be the common method in coal mining, and more particularly in coal mining than in metal mining. It seems to me, although it may be of value in coal mining, it is of less value in metal mining, because of varied conditions of working in different parts of the mine, different methods employed in different stopes, and different methods of timbering employed under varied conditions of ground, peculiar conditions of working in mines which have been developed, and have many old workings.

I am able to quote from some figures obtained at the Franklin mine during the past three years on the same basis, although it does not seem to me such figures are of as vital importance as those based on actual amount of time worked. In 1912, which was before either of these bonus systems was installed, the rate of fatalities was about 12.5 per million tons of ore and rock mined. During the year 1913, in eight months of which the lump-sum bonus system was installed, the rating per million tons was slightly less than 4, dropping from 12.5 to 4, and during 1913, in which the improved bonus system was in vogue, the rate of fatalities dropped to 1.3 per million tons of ore and rock mined.

Now, the fatalities, as previously noted, are often a question of good or poor luck. The system in vogue does not pay particular attention to fatalities, but takes into account any accident which is anything more

than a very minor accident—any accident which requires a loss of time is of very great importance in the acquirement of a bonus. Based on the United States rating of a serious accident, one requiring more than 20 days of time lost, during 1912 there were 54.2 per million tons mined. During 1913 there were 26.6, and during 1914 there were 24.9, which shows that the serious accidents dropped to about one-half. The slight accidents, which required a loss of time of more than one day, and not over 20 days, dropped from the figure of 690 per million tons mined in 1912 to 255 in 1913, and 156 in 1914, which gives a good idea that all of the accidents have been greatly improved by this method of bonus payment, and I cannot emphasize too strongly my personal conviction that the bonus system of payment to shift bosses or under foremen is of great value in the reduction of accidents.

It is only natural that a man impressed with getting out a high tonnage, and seeing that conditions are generally safe, is apt not to take the time to caution a man about minor things, things which are perhaps occurring daily, because he feels that anybody with any common sense would know better than to do those things. He has probably told the man a dozen times already not to handle a bar or chute in that particular manner, and thinks if the workman continues to do it that way he deserves to be injured. But pay him a bonus, which amounts to 10 per cent. of his monthly wages, to avoid these accidents and it is giving him an opportunity to place this money in his pocket by being a kindergarten teacher and seeing that these men are instructed against the minor potentialities for accident.

William H. Grady.—In the discussion some doubt was expressed as to whether or not systematic timbering was superior to non-systematic timbering in the matter of accident prevention. The point of view stated was that systematic timbering is designed to take care of ordinary conditions of top and that when extraordinary conditions of top are encountered the men may be less apt to guard against them, depending for safety on the systematic timbering.

My observations are that most accidents come from the ordinary conditions of top. For example, it is generally conceded that fewer accidents occur in the robbing than in the room work, although the top conditions are admittedly much worse.

The United States Coal & Coke Co., as I understand it, makes a practice of timbering systematically, even though the roof is good, to the extent the Advisory Board deems necessary to guard against ordinary conditions of bad top. The matter of guarding against extraordinary conditions of bad top is left to the judgment of the assistant mine foremen, checked up by the mine foremen and timber inspectors.

In non-systematic timbering the matter of taking care of all conditions

of top is left to the judgment or opinion of men, some of whom are recent arrivals from southern Europe who would not know a "kettle bottom" from a keg of powder. By systematic timbering ordinary conditions of bad top are provided for, and extraordinary cases of bad top are taken care of by an assistant mine foreman, who is employed because he has shown ability and proficiency in judging and acting in such matters. But that is a question of opinion as against opinion, and in the absence of facts we must depend upon opinions. However, when we have facts and statistics on a subject we should observe the facts and study the statistics. I have no doubt that a careful study of the facts and statistics, as presented by Mr. Eavenson, will lead to the conclusion that, in so far as the prevention of accidents is concerned, it does pay to timber systematically and exercise a high degree of supervision. From my perusal of Mr. Eavenson's paper, I am in doubt as to whether or not sufficient statistical data have been presented to form a conclusion in regard to the cost of systematic timbering. It is my understanding that the increase in the efficiency of the men and equipment, the more economical use of material, and the greater regularity with which the absence of slate falls permits the working of a room has more than offset the additional cost of timbering and supervision.

George S. Rice, Pittsburgh, Pa.—A previous speaker, after commending the plan of the U. S. Bureau of Mines in taking moving pictures, for educational purposes, of good and bad practices in coal mining, regretted that it had not carried on a similar campaign in metal mines. I am glad to state that the Bureau has during this past year taken moving pictures in many metal mines and quarries throughout the country. While the set has not been completed, the Bureau will ultimately have reels of many scenes quite as good as the first ones taken by the Bureau photographers in co-operation with the United States Coal & Coke Co. in the Gary coal mines.

In regard to the question of systematic timbering as a measure of safety, I have been very much impressed with the advantage, as shown very strikingly in foreign practice, of systematic timbering; that is, fixed or regular spacing of props and other timbers, regardless of how unnecessary it may appear to the miner or to the foreman. It seems to me that this feature represents one of the greatest differences between the safety practices followed in the coal mines in Europe and the coal mines in this country. I refer particularly to the good results obtained by close, regular timbering in the mines of Pas-de-Calais district, France. Their record for small loss of life from falls of roof is very remarkable. As some of you know, there they timber without regard to the appearance of the roof, and have a most elaborate system of supporting the roof. It is expensive, but so far as the lessened number of accidents is concerned,

it certainly pays, and I think it does also from a business standpoint. Roof and coal falls cause almost half the loss of life in American mines.

I was impressed by the features surrounding an accident, given in a report which recently came into my office from one of the Bureau engineers, in which a number of men had been killed by a large fall of roof. This fall had occurred on a main entry, and the report stated there had been no previous indication of any weakness in this roof as tested either by the sounding method or the more approved method of touching the roof to feel if there is any vibration while the roof is being smartly rapped with a bar. It indicated no weakness because there was such a large mass; but, when the fall occurred, it was found that there were slips parallel with the ribs of the entry or gangway which came up in a more or less inverted V-shape; the wedge-shaped mass had been supported by narrow lips resting on the edges of the ribs, on either side of the entry. The mass had thus been resting on insignificant supports, and when these weakened, it permitted the whole mass to fall on the men. That shows a case where, perhaps, systematic timbering, regardless of previous safe appearance of the roof, might have saved those men.

Arthur Hovey Storrs, Scranton, Pa.—In regard to timbering, I think that not only should the timbering be done systematically, but use should be made of the foreign practice of "collars," or double timbering, instead of the single prop, which is responsible for much of the decrease in falls of roof. The caps or collars there used are very light, but large enough to catch the small slabs of roof which fall and cause so many of our minor accidents.

We have not in this country the autocratic power to stop the use of alcoholic drinks, which we have been told to-day has been used in other countries, but the efforts of one company in the anthracite field may be of interest. The Delaware, Lackawanna & Western Coal Co. in a part of its field has made an appeal to the saloon keepers not to open in the morning until after seven o'clock, so that the men may go to their work without being tempted to drink liquor on the way. The request has met with considerable opposition from some of the saloon men and how it will work out finally I do not know. The coal people are hoping that the courts, in granting the next licenses to saloon keepers, will make it a rule, in connection with the license, that the saloons must be closed from midnight until seven o'clock in the morning.

The Lackawanna company also, some time ago, brought out what might be called a primer on accidents, in which photographs of proper and improper methods of doing certain work were shown. These were printed side by side and the things which should not be done were printed in red ink, as a sort of danger sign. The company also used this booklet as a primer to teach the foreigners the ordinary English, used about the

mines, without attempting to make correct English sentences. That, I think, has been of considerable value.

I hope I may be pardoned for citing one instance in connection with a visit to Mr. Eavenson's mines, which I ran into, as showing that their officials do not simply make the rules, and let it stop at that. I went through the mines with the General Superintendent, Mr. O'Toole, and in one room a small fall of rock had knocked one prop out of place. In replacing it the assistant foreman had set it on top of a fallen slab of slate. The assistant foreman was not present at the moment, but meeting him two or three rooms farther on, he was at once given instructions by Mr. O'Toole to go back immediately and reset the prop properly, removing the fallen slate.

HARRY H. STOEK, Urbana, Ill.—Several of the speakers have referred to the use of lantern slides and moving pictures. The Committee on Junior Members and Affiliated Student Societies has been trying to get together during the past year a list of such slides and films. While the committee has obtained this information primarily for the use of the colleges and universities, it has occurred to me during the discussion this morning that they might be made of very much more general value. Mr. Goodale stated that his company had such a film. A number of mining companies run night schools for their employees, and while the larger mining companies can afford to get moving pictures, the small ones cannot afford it. If we can get a list of all available material of this kind and publish it in the *Bulletin*, and can arrange some exchange system by which the different companies could use this material, it might be of advantage, not only to us who are teachers, but possibly to the mining companies as well.

As to the practical value of these pictures, I would cite an instance that occurred recently in connection with the miners and mechanics' institutes of Illinois. Films obtained from the U. S. Bureau of Mines were shown each night at a different mining camp in Illinois through an arrangement with the moving-picture places in the camps, by which these films were shown after the regular performance, or in one or two places as a part of the regular performance. After one of the performances a man came up and said: "I saw that same film some few months ago, and the next day, when we went into the mine, and I started to do something in a wrong way, my 'buddy,' who was a foreigner, said, 'Don't do that, the pictures said not to.' "

LAWSON BLENKINSOPP, Landgraff, W. Va. (communication to the Secretary*).—During the two years I have been in this district as State Mine Inspector, I have taken much interest in the advancement of the

* Received Feb. 14, 1915.

safety-first method. Nowhere in the 11th district has such a move been made to discipline and organize for the benefit of the workmen as at the United States Coal & Coke Co.'s plants. Several points make this a difficult undertaking: First, they have very raw labor, and have to educate them; second, the place does not command men, as it is on a branch line isolated from the main line; third, men are prone to be antagonistic, at first, to safety methods, but they soon learn it is for their welfare, and become an interested party to the organization. The employee has a duty to perform and he is given to understand this, when he is employed. A book of rules and the mining law are given him in his language, and he is further instructed from time to time. The rules are enforced by the assistant mine foreman, who visits each working place three to four times daily and stays until his orders are complied with. Each employee is rated at a certain capacity, and should his efficiency fall below this standard, after a reasonable trial, he is dropped. This, in my opinion, is one of the surest methods to overcome the complaints of the miner, that he goes into the mine from day to day and cannot get cars to load his coal, that he cannot make a living, etc. At the United States Coal & Coke Co.'s plants it is the reverse; it is up to the miner to load each day the amount that is cut for him; each district assistant foreman knows how much coal he can get each day by 8 a.m. and that amount must be forthcoming; also each place must be cleaned up by quitting time, so coal can be cut again that night, unless a reasonable excuse is given.

I wish to make special note here as to roof conditions. In nearly all the mines roof conditions are dangerous. Of the 18 operations only five have good roof. The Pocahontas No. 4 seam has a rash following down from 3 to 5 ft. thick, and is dangerous in the extreme, requiring careful timbering. Systematic timbering is enforced in all conditions of roof, good and bad alike. Timber is removed from good roof places and used over again as the pillar is extracted. Many of the mines operating the Pocahontas No. 3 seam have a draw slate 30 in. thick. This is above the average thickness of slate in mines operating on the main line of the Norfolk & Western Railway. When roof conditions become dangerous mining is stopped, the place is timbered, or the loose roof taken down. They do not wait until the run is over, but do it at once. Nearly all operations have their own method of working. The latest economic method of the company is to extract advancing, and while the excavating is fresh pull all pillars, thereby eliminating the dangers due to crushed pillars, broken roof and brows of disintegrated draw slate. Many accidents are due to men going back too soon after shooting, without regard to the condition of the roof. This cannot be done at the mines of the United States Coal & Coke Co., as the assistant foreman does all the shooting in his district; after the shooting, the miner is not allowed to go back until his place is examined by the assistant foreman.

Ventilation is all that could be desired in a coal mine and the fixing of a safe minimum is an excellent regulation. It obligates the assistant foreman to keep all brattices and stoppings in his district in good condition, and should the company inspector find less than 12,000 cu. ft. at any last breakthrough the assistant foreman is called to account.

The premium system is good as it causes "competition safety;" also the posting of the accident list at each mine, showing minor, serious, and fatal accidents, is helpful. The official not interested in reducing accidents is not long-lived at this company's works. The weekly meetings of the officials are to discuss the prevention of accidents, by endeavoring to find the cause of any accident that may have occurred. A typewritten statement is posted on the bulletin board at face of each room, showing the cause of every accident that occurs and the method of preventing a similar one. The exhibition of moving pictures showing the various mining operations has been helpful in impressing on the miner's mind the dangers surrounding his work.

In justice to the other operators in my district, I would say that all have co-operated heartily in advancing the safety movement. Nearly all the mines have eliminated the dust problem; solid shooting has been practically eliminated. Clay is being used at most mines for tamping. Sanitary conditions have been improved, some operators providing wash houses for their men. Safety inspectors have been employed to aid in the prevention of accidents. In fact, many of the companies in the district are running a close second to the U. S. Coal & Coke Co. in the matter of safety.

Howard N. Eavenson.—In reply to Mr. Goodale's question about the assistants and the average number of places attended by each, we do not have that uniform throughout all of our mines. The idea is to make it as nearly uniform as possible, but the number of men whom each assistant oversees is regulated also by the distance which the assistant foreman must travel.

In addition to having a regular section in the mine to oversee, each assistant has in his charge a certain portion of the main-haulage roads and manways through which the men must travel to reach their working places, and he is responsible for any injuries occurring in these places. The assistant with a small number of men will have a larger section of haulage headings and manways than the one with a large number of men.

As to systematic timbering, we have a number of places, particularly in No. 3 seam, where portions of the draw slate over the coal are in the shape of truncated cones. Sometimes these "kettle bottoms" are only 2 or 3 ft. in diameter on the under side, and a foot or two on the upper side, and sometimes they run to 10 or 12 ft. in diameter, and it is almost

impossible, when the place is first excavated, to discover them. The systematic timbering, in designing the layout of the rooms, is arranged so that each timber supports a cap piece, such as Mr. Storrs mentioned, which is likely to catch these kettle bottoms, and the posts are not over 5.5 ft. apart; at least, there is one every 5.5 ft. It is our standard to drive double-track rooms and right along each track we have a row of posts 3 ft. apart with the caps extending at right angles to the track, the theory being that they will protect the roadway to some extent, while in the remaining portions of the room the caps run parallel to the roadway itself. The timbers are kept not farther than 6 ft. from the face at all times; as soon as a portion of the coal is excavated the posts are set, and they are kept to the face at all times.

Replying to Mr. Goodale's other question about the efficiency having decreased, the figures of number of men employed, taken from the diagram in the table, include all outside men, and there were a great many of them on construction work in 1913, and the efficiency curve derived from those figures, as far as inside work is concerned, would not be strictly correct.

In 1911, by some time studies on the amount of time that was spent by the miners in loading, laying track, setting timbers for taking care of the roof, etc., and waiting on cars, we found that they were not working over 40 per cent. of the time. At that time our men were averaging about 2.5 to 3 cars apiece. By rearranging the haulage system and putting on a few more drivers, so that there would be no excuse for the miners waiting on cars, we were able, without increasing our piece rates at all, to increase the average earnings of the men $1 a day. In other words, we increased the output from about 3 cars to about 4.5 cars per man per day, and for the year 1914, the whole year, our men averaged 16.4 tons of coal per man per day at the working face. That does not include the drivers, or the machine men, but just the miners themselves.

Regarding the reduction in accidents shown, basing our figures on 1909, which was the first year we began this work, and taking the tonnage per injury of that year as a standard, I have the figures for the fatal accidents and serious accidents. We have reduced the fatal injuries 70.7 per cent., and the serious injuries 46.1 per cent. I have not the figures here for the minor injuries, but I think they would show at least a corresponding reduction. Last year we produced 458,900 net tons of coal per fatal injury inside the mine, and 367,100 tons per fatal injury, both inside and outside. In Mr. Dawson's discussion, he did not mention the fact that the H. C. Frick Coke Co. in the same year produced 977,000 tons of coal per fatality inside.

I have the figures for Great Britain, for 1913, the last available official figures taken from the inspectors' reports, and in that year they produced in Great Britain, including all the mines in England, Scotland,

and Wales, 207,000 tons of coal per fatality inside. Taking all injuries, fatal, serious, and minor, and including everything from which a man lost over seven days' time, we mined 14,340 tons per injury in which any time over seven days was lost. I do not know how that compares with all of the foreign countries, but it is about ten times what the English figures show.

ARTHUR WILLIAMS, New York, N. Y.—Some figures presented to the American Museum of Safety a year ago showed that the average fatal accidents over a period of five years in European mining was one to about 168,000 tons of coal mined, and the average in this country for the same period was one fatal accident for about 186,000 tons of coal mined, taking the country as a whole, and including such States as Alabama and Oklahoma and British Columbia, in coal mining.

I understand that the German manufacturers have found that where they provide soft drinks for their men, in the absence of any municipal regulations, the tendency to drink alcohol disappears, or is greatly lessened, the men finding the soft drinks a good substitute.

G. S. RICE.—I think the movement is splendid, but we do not want to flatter ourselves too much by making the accident rating on the basis of tonnage, because there is a most important factor in the ease or difficulty with which we can get the coal. For example, in the longwall district of northern Illinois, the 1913 figures show that the daily output per employee is but 1.96 tons, whereas in Williamson County, in the southern part of the State, it is 4.06 tons per man. The miners are chiefly foreigners in both places; they interchange and belong to the same union; in other words, are just the same kind of men. The difference in the output is because in the northern part of the State the seam is thin, and much rock must be hoisted, and otherwise a great deal of dead work must be performed in longwall mining, whereas in the southern part of the State the dead work is comparatively light, and the coal is largely cut by machine. Again, in Pennsylvania I see that in 1913 the soft-coal mine employees averaged 3.78 tons per working day, while the anthracite men averaged but 2.02 tons of 2,000 lb. per man. Would any anthracite operator or miner admit that the anthracite men were less competent, or the mines were less well equipped or administered than the bituminous mines of the State? I think not, and it is for similar reasons I believe it is a mistake to compare the number of individual accidents per thousand tons of coal produced in one district, State, or country with that of another district in which the conditions may differ widely. One might almost as well compare the accident rates in copper mines of the country as a whole on the basis of net tons of copper produced, which would be less a comparison of relative risk of life or limb than of percentage of copper in the ground.

In making comparison with European countries, both of accident rates on a tonnage basis and of costs of producing coal, we must take into consideration that in general the seams abroad are steep pitching and badly faulted; that machine mining can rarely be done; the work is deep and the gaseous conditions more serious than in this country except in our anthracite district. The seams are thin, and even when thick the advantage of cheap mining is lost to a great extent because the operators are required to pack or stow the excavations either by hand or hydraulically, to prevent excessive subsidence of the surface: furthermore, the mining of the thin seams, which may be done at monetary loss because of the dead work, is carried on simultaneously from the same shafts as the thicker seams, which decreases the average yield per man, and increases the average cost of the output.

While the method of comparing annual individual accident rates on the basis of 1,000 employed is open to some objections, it better enables one to consider the relative risk of the different kinds of employment in and about any one mine, and also of different mines, with the same or with other conditions. To better compare the actual risks, it seems advisable to standardize the relative exposure on the basis of time; that is, if the men at one mine, on an average work 200 days of 10 hr. in a year, and in another mine 300 days of 8 hr., the men of the first mine would be exposed for 2,000 hr. during the year, and in the second mine for 2,400 hr.; the men in the second mine would have had additional exposure in the ratio of 24 to 20. A standard year of 2,400 hr. might be selected and the rates figured to this basis. All mines keep account of the number of hours run per day, so that there would be no difficulty in reporting the hours per year, and figuring accident rates on that basis.

Charles W. Goodale.—In reply to Mr. Rice, I will say that we have adopted in the mines of Montana the basis of so many accidents per 10,000 shifts, for the reason that every shift represents a risk, which may be incurred either in development work which has to be done, or in the breaking of ore.

Reviewing what we have done in the "safety first" work, and comparing 1914 with 1913, I will say that we have reduced the fatal accidents 35 per cent. on the basis of 10,000 shifts, and the serious accidents 26 per cent. In the case of minor accidents, we have not been so successful —we have only reduced them about 3 per cent. We have adopted the plan of getting the shift bosses together every month and making them stand up and tell how each accident under their jurisdiction occurred, and every one present is then called upon to make suggestions as to how that kind of an accident may be avoided in the future.

Howard N. Eavenson (communication to the Secretary*).—From the tenor of the discussion of this paper, as well as remarks made to the

* Received Mar. 18, 1915.

writer by various members since the meeting, the general opinion seems to favor the system of giving bonuses to foremen for work of this character and, apparently, the general opinion is that all men engaged in the handling of labor should be in favor of a system of this kind. This, however, is not the case, as some take the view that where a foreman is paid to do a certain thing it should not be necessary to offer him additional inducements in the way of bonuses for performing what is apparently a portion of his work. This may be true if the foreman is receiving a fair compensation for his labor, but the answer to this point of view is that the attitude of the public within the last six or seven years has entirely changed with reference to industrial injuries. Under the old theory the workman was supposed to assume all risks of the employment and could only recover damages for injuries received when negligence on the part of the employer could be shown, and under this theory safety measures that were taken were adopted purely from a humanitarian point of view. The new theory, now almost universally held, is that the business in which any workman is engaged should be responsible for the cost of all injuries received and that the cost of all such injuries is a legitimate charge against the cost of production; and that compensation should be made for all injuries, no matter how occurring. In accordance with this attitude, employers must use all known safety devices so there will be fewer injuries to the workmen and thus necessarily lower the cost of production. In addition to the humanitarian side, the employer now finds it a commercial proposition to keep his works or mines as safe as they can possibly be, not only for the sake of the money saved in the compensation claims, but also from the increased efficiency and better results achieved in all lines from the men when working under safe conditions.

In all lines of endeavor this same change in attitude can be seen. The old theory was that the employee must follow instructions or be made to suffer for not doing so by being reduced to an inferior position, suspended for a definite period, or discharged. The new theory differs from this in the fact that ordinary services receive the regular rate of compensation, services above the ordinary an extra rate, and the failure to perform ordinary services is treated as rather due to incompetency or inability than to neglect, and in any of our large establishments it is customary to find a workman changed from one position to another until he at last finds work he can do satisfactorily, or is finally discharged from the rolls.

All energetic management to-day is trying not only to educate the workmen in safety measures but in efficiency, sobriety, and general industry.

Enlarging the Worth of the Worker and the Perspective of the Employer

BY J. PARKE CHANNING,* NEW YORK, N. Y.

(New York Meeting, February, 1915)

THESE days of great industrial and social problems in America produce many suggested solutions and great changes. The practical engineer and employer of labor views these problems differently from the labor leader or the social reformer, but as never before he is sincerely interested in solving them in a way that will be just to all.

The inevitable tendency of the day is toward "industrial betterment," "safety," "industrial education," "efficiency," and the many other things which have become so familiar to progressive employers. There is no longer any question that these things are worth while from both the human and economic standpoints. They "pay" in dollars and cents.

The very center of final success in improving conditions and increasing the efficiency of workingmen must be the spirit of fairness and a knowledge on the part of the employer of how to deal *sympathetically* and *intelligently* with his employees. Every progressive employer knows how greatly he desires foremen, superintendents, managers and others who possess these qualities. On the other hand, we are all familiar with serious mistakes made by young graduates of engineering schools who have had no opportunity to develop these qualities, and who have no real appreciation of the worth of the workers. Indeed, one wonders whether much ill-feeling, labor difficulties, and many strikes could not be avoided if such men had the right attitude.

Is there any way of remedying this condition? If this particular difficulty can be solved, if these young engineers, many of whom are our coming leaders of industry, can be given the right perspective and the right understanding of these other problems in addition to fair, sympathetic methods of handling men, many of our other problems will be solved—not at once, but gradually and permanently, as these men make good and become influential in paths of industrial righteousness and industrial peace. Many progressive employers of to-day have enlarged their own perspective and realize the great importance of enlarging the perspective of those who shall follow them.

How can it be done? For seven years a movement has been making

* Vice-President, Miami Copper Co.

rapid progress in engineering schools with the purpose of helping to solve this very problem. It was started at Yale in 1907, by the Young Men's Christian Association, when some engineering students were led to get in touch with workingmen and boys in New Haven. The idea was to render service by teaching them English and other subjects and in turn to learn their ways, ideas, customs, and how to deal with them intelligently. Friendly, mutually helpful personal contact was the basic principle. This was the beginning. Do not confuse it with "social service"—it was this, and much more. The reaction on the engineer was the main object sought. The idea worked out so successfully that a number of men saw great possibilities in it, and the whole conception was greatly enlarged. Under the name of the Industrial Service Movement, it has spread to 200 other colleges and technical schools in the past seven years, and has justified itself from every point of view. It is really helping in a vital way to solve the special problem we have been discussing and other problems as well. It is, to put it briefly:

Plan: Bringing engineering students and industrial workers together to their mutual understanding and their mutual good.

Purpose: To get workingmen educated and educated men to work. To send men out of college with a new sympathy, a new vision and a new determination to help.

Principle: Fraternity—not to go down to help others or to ask others to come up and be helped, but rather to go *with* them, not in any sentimental way, but in a spirit of common-sense brotherhood.

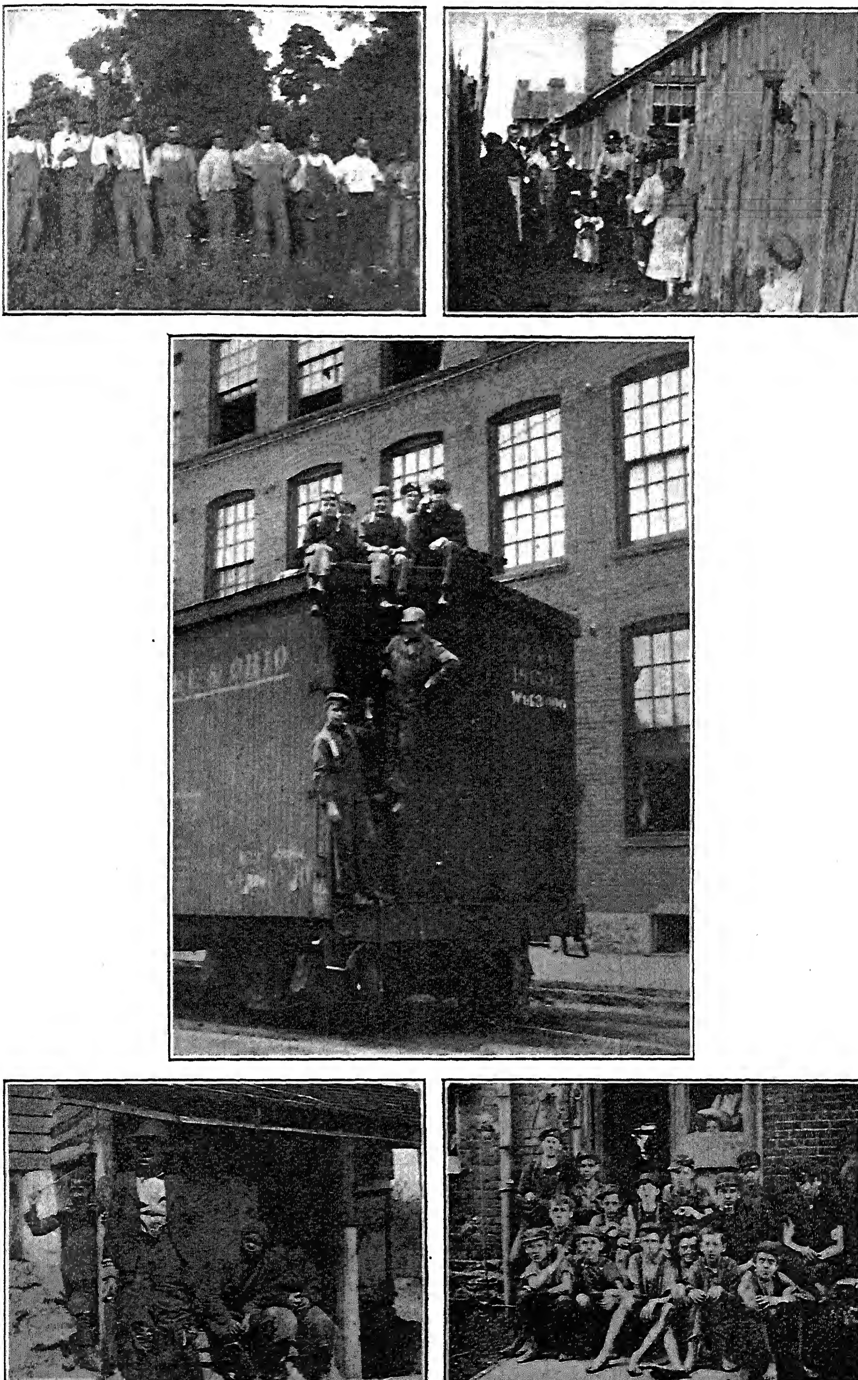
Method: Putting college students up against real opportunities for the kind of service which appeals to them, such as teaching foreigners English and citizenship; instructing American workingmen in technical subjects; leading clubs of working boys, etc. There is opportunity for every leader's peculiar ability to assert itself, in any way that is real. Other methods will be described later.

Accomplishment: During the past year 3,500 students from 200 colleges have engaged regularly in industrial service; 3,000 graduates are active in industrial betterment as a result of interest acquired while at college, during the past seven years.

Leadership: The Young Men's Christian Association, through local branches, State committees and the industrial and student departments of the International Committee.

Co-operation: The Movement works locally through the Young Men's Christian Associations and any other recognized agencies for industrial and social betterment in the community. Professors and students, employers and employees, engineers and social workers heartily co-operate.

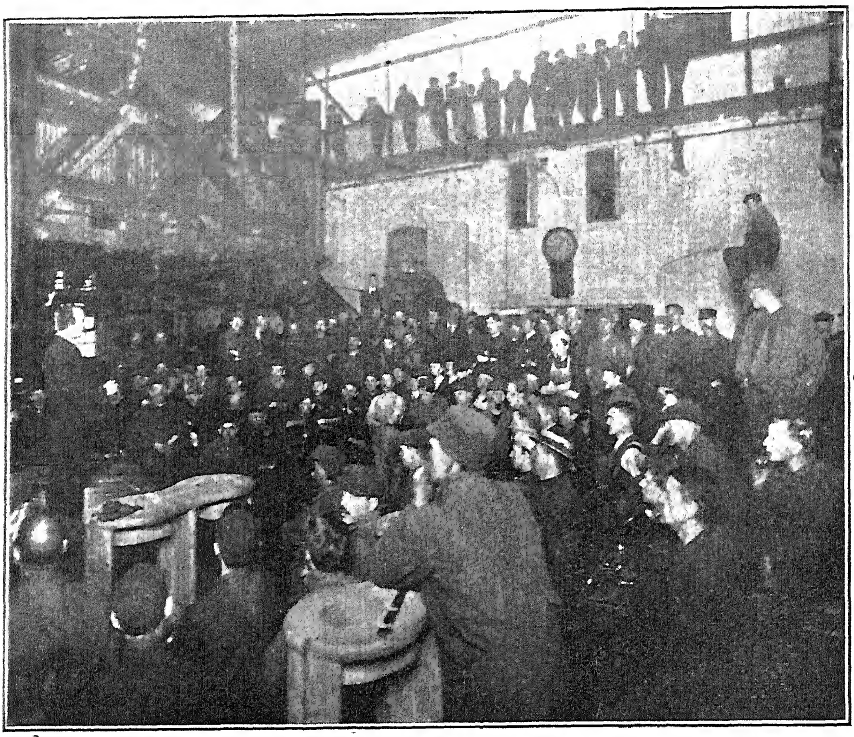
Significance: Experience proves that men interested in this work at college go out into the larger world with a new vision and a new attitude and sense of responsibility. These men will largely determine whether



WANTED: LEADERSHIP.

conditions shall be good or bad and whether the human factor will be given fair consideration. How better can the problems of capital and labor be solved than by mutuality, good will, efficiency and character in business? The nation's hope is in the coming leaders who shall follow us and who possess such essential qualities of success. The development of such leaders, with their continually increasing capacity for service, is the ultimate purpose of the Industrial Service Movement.

It may seem surprising that 3,500 engineering students, each carrying a heavy course of study and with many other interests, can find an evening



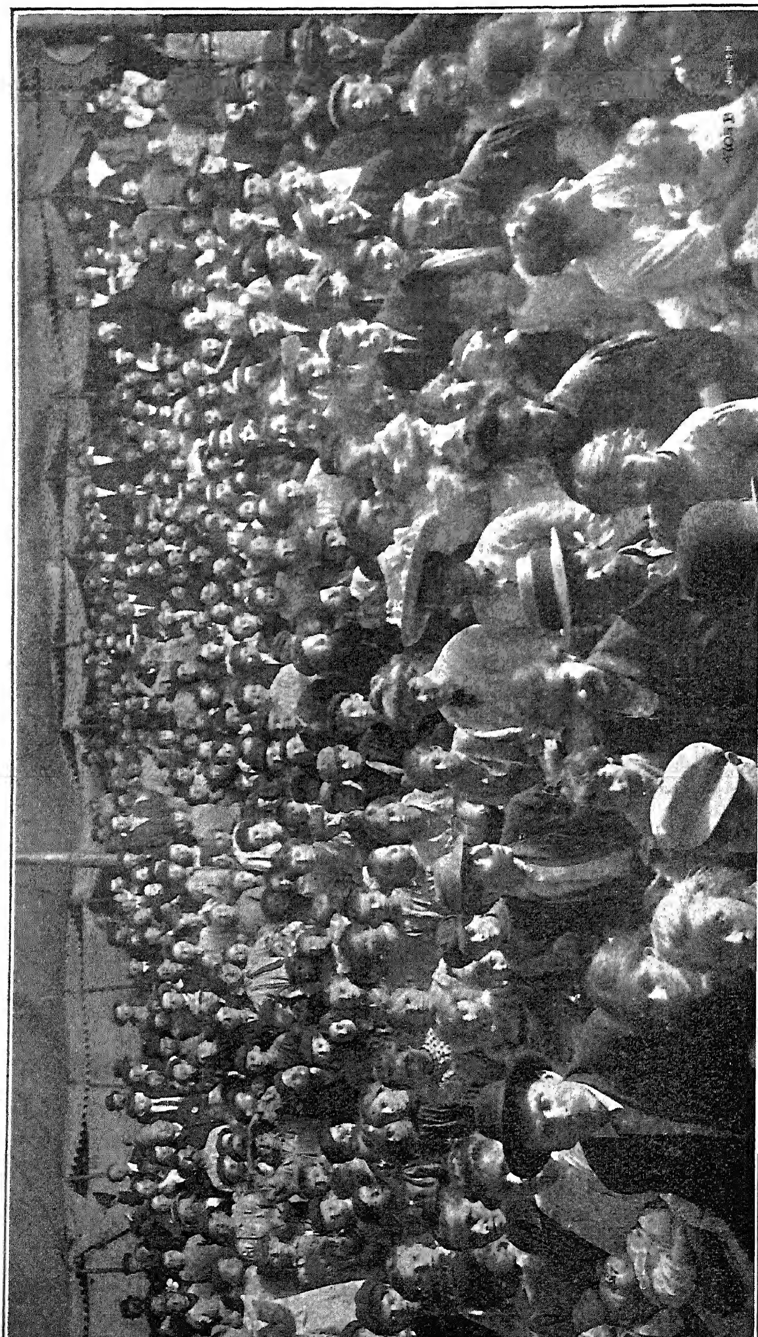
SHOP MEETING.

or two each week to engage in some form of definite service, without any financial compensation. But such is the case, and on the whole a careful survey of their work reveals efficiency and permanency in a high degree. If industrial men are at first suspicious, their suspicion soon wears away in the face of frankness and friendliness. If the employer has any doubts, they do not last long. One may travel around the country and observe students teaching foreigners in railroad box cars, stores, clubs, halls, pool rooms, restaurants, and boarding houses as well as in the more dignified meeting places—schools, churches, settlement houses, and factories. One may see American workingmen instructed in mines, shops, and labor-

union headquarters. On may look with interest upon recreative games, talks, first-aid and safety promotion in all sorts of places at noon, afternoon, evening, and midnight. And one may see 500 men crowded around the machinery of a huge plant listening to a straight noon-hour talk on clean living, character-building, and vital religion. We have looked with amazement on 50 factory boys following enthusiastically a college football captain who took enough interest in them to organize a boys' club or a factory athletic league. It has all been done in the finest kind of spirit, without patronage, with modesty and with efficiency. And during the past year those 3,500 student leaders reached over 60,000 workingmen and boys in a very personal and directly helpful way. The Secretary of this Movement has talked with hundreds of employers and college professors throughout the country and all seem enthusiastic over what has been accomplished.

But what has this to do with engineering? Just this—that every one of these 3,500 students would be willing to say that he has gained far more than he has given. Furthermore, a study of the situation proves that he has gained in large measure the very qualities he needs—an appreciation of workingmen, adaptability, leadership, a knowledge of how to deal with men in a way to get results and to avoid harmful labor difficulties. In general, he learns that all men are men, regardless of race, nationality, color, or creed, but that men must be dealt with very differently; he learns that it pays to win the leaders of men if one desires to win the men themselves; that the work, home and leisure life of industrial workers play a large part in determining efficiency; that a man's working associates may largely influence the quality of work he does; that helping men to concentrate on their work (though not at the expense of mental and physical welfare) increases output; that friendly competition (without driving men) helps break records; that reasonable relaxation and recreation pays both from the human and economic standpoints; that visitation of other plants and stimulation of new ideas in various ways may mean a money saving to the company; that loyalty of the men is one of the employer's greatest assets; and that character counts most of all. More than this, he learns to understand men, he learns how to sympathize with the other fellow's point of view and how to handle men successfully. Is this not worth while? Who can foresee what the future will hold for these men in the way of tremendous opportunities and responsibilities?

Let us illustrate. One engineering student apparently never took any interest in any one but himself until he was enlisted in some of this work. Two evenings each week he walked two miles to teach a class of 20 coal miners. The miners learned a great deal, but they little realized how much they were teaching the college man. His whole viewpoint was gradually changed. He learned to appreciate that all men were *men*, and he graduated from college with a new vision and a new sense of responsi-



HEALTH TALK TO PITTSBURGH STEEL WORKERS AND FAMILIES.

bility. He had not become a sentimental idealist. He perhaps realized the weaknesses of workingmen better than ever, but he had come to know their good points as well, and he had developed a real point of contact. It was therefore not surprising to receive a letter from him recently, indicating his growing interest and telling enthusiastically of his success with a welfare club house and other educational, recreative, and social features introduced for his miners.

It happens that I am a member of the Advisory Committee of the Industrial Service Movement, the headquarters of which is at the office of the Y.M.C.A. International Committee, 124 East 28th Street, New York. Several times I have seen letters from recent engineering graduates, which have come to the central office. These letters tell the story in no uncertain terms, and from them the following quotations are taken:

"I have organized several classes for the men in our plant. I can say honestly that this friendly basis with my men helps rather than hurts discipline. The work is but a beginning of what my company hopes to do."

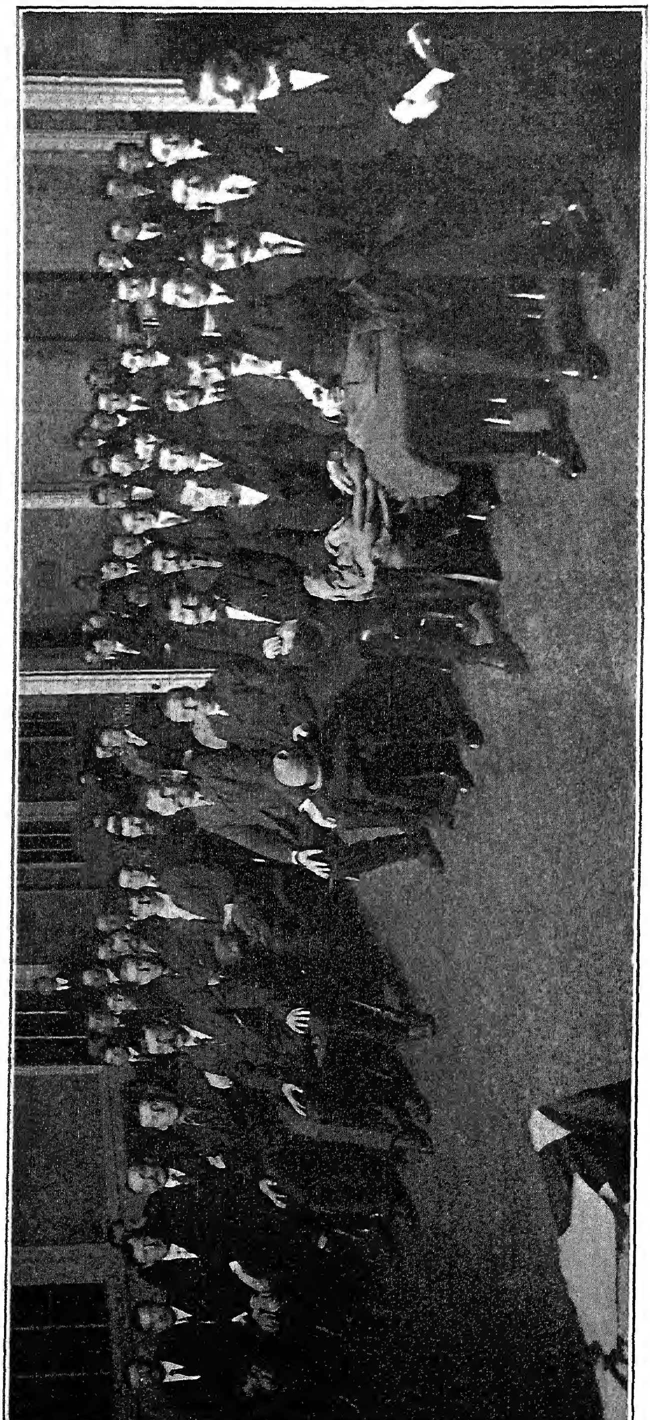
"You will no doubt remember that I took up this work last winter at the University. Now that I have gotten into the habit I really like it so much that I am devoting some of my time to it now, though I am very busy with my business. I have gathered together a class of about forty Italians and enjoy it immensely."

"My company is just now organizing a scheme whereby classes will be offered to young men in such subjects as relate to their work. I have consented to teach one of these groups, as this is the same sort of work I did when at college."

"I have read the literature with care and interest. I expect in the near future to be called to a position in New Mexico where I will come into close contact with foreign miners. My special work will be in the promotion of education and accident prevention."

"I consider the contact that I had with the Industrial Service Movement the most valuable experience in my undergraduate days. I would never be where I am to-day, without it."

There are numerous other ways in which the Industrial Service Movement is making itself felt in the colleges. These can only be mentioned briefly. Special lectures on the human side of engineering and kindred topics are given by selected employers, engineers, and social workers. These give the students a vision which does not come from most technical courses. Some colleges have promoted strong two-day conferences on safety and industrial betterment. The student engineering societies are stimulated to have such lectures and to take up the practical aspects of the work. Weekly discussion groups are held for interested students, an industrial reference library is installed, and a live bulletin board keeps the work before the student body. As senior engineering trips pass through New York, Chicago, or other cities where the Movement is strong, professors in charge frequently arrange for a day to be spent in studying "the human side" and the work of various welfare agencies. One of the most significant recent developments is the proposed course on the "Hu-



Class in American Citizenship.

man Side of Engineering," which will be hereafter included in the regular engineering curriculum of some of the largest colleges. This course will include such topics as: "The Human Factor in Industry," "Historical—The Evolution of the Individual Worker," "Industrial Organization," "Influence of the Modern Factory System," "Evil Conditions which Retard Progress," "Efficiency Conditions," "The Welfare Programs of Typical Companies," "Industrial Betterment," "Scientific Management on its Human Side," "How to Handle Men," and "The Engineer's Responsibility for Service."

Many articles about the Movement have appeared in engineering and other magazines. Some of the national engineering societies are very much interested in the whole plan, and many points of co-operation between the American Institute of Mining Engineers and this Movement will at once be apparent.

There is another side of this whole question which must not be forgotten. The Movement stimulates great interest in the colleges. While 3,500 students are actually engaged in service, many thousands of other students and professors hear the Movement presented and become heartily interested in its ideas and ideals. Is there danger of this enthusiasm not being conserved after graduation?

The Young Men's Christian Association recognizes this very danger, and willingly places its world-wide organization of over 8,000 branches at the disposal of these engineers who have become interested in the service idea. Whether the graduate engineer goes to the frontier or into the industrial city, this Association is an instrument ready for his use. In the larger cities it is being used to perform definite pieces of service which the industry may need. Industrial and immigration secretaries are available to co-operate. For example: A recent graduate entered the employ of the Ford Motor Car Co., in Detroit, and at once manifested his interest in the foreign employees. The company officials noted his enthusiastic interest, and used him in their enlarging plans for educating their men. He naturally turned to the Young Men's Christian Association, which had prompted his interest while at college, and brought the Association's special methods of work with foreigners to the attention of the production manager. The result is that over 1,100 men in that plant are now learning English, and American citizenship, under the auspices of the Association. Special lessons were prepared for this industry. This is being done also for the mining industry, and accident prevention is taught with the lessons by graphic pictures. In the remote engineering fields this co-operative agency is quite as valuable as in mining and lumber centers and in construction and reclamation camps. Thus the machinery is available, and members of this Institute will be especially interested in what some mining men have said of it.

Dr. J. A. Holmes, of the U. S. Bureau of Mines, says:

"I am interested in the Y M.C.A. because during the last twenty-five years, as I have passed up and down through the great mining regions, I have seen places transformed through its work."

At a dinner recently, John Hays Hammond made the remarkable statement:

"This work is one of the most promising things that I have yet seen in connection with modern industry."

Judge Elbert H. Gary, Chairman, Board of Directors, United States Steel Corporation, has said:

"I am glad to declare my belief in the advantages of having a Young Men's Christian Association in an industrial community as tending greatly to the building up of the character of the men, and therefore increasing their efficiency."

It is needless to say how much I am personally interested in the movement. The principle is a sound one, and I can already, from my own experience, see the good effect it has not only upon the young men who are engaged in this work but upon the men to whom they are devoting their energies.

Any such agency which can improve the educational, physical, social, and spiritual welfare of employees is bound to increase their efficiency. Fred H. Rindge, Jr., Secretary of the Movement, writing in the *Mining and Scientific Press* of Nov. 7, 1914, said:

"Practical money-making industrial leaders (even the unsentimental ones) have found that such a factor in modern industry develops trustworthiness, higher standards of workmanship, and makes possible that kind of intercourse between employers and employees which begets good will, and good will raises all other forces in industry to a higher power. The right kind of an investment in men will yield big returns."

The whole plan seems to me sane, practical, and convincing. It helps the coming engineer to get a broader perspective, while an undergraduate. It enlarges the worth of the workers through the service of the students. It affords an instrument for all-round welfare work wherever the engineer may go. I am reading this paper because I believe our Institute will want to know more of this work and because of the direct opportunities for helpful co-operation through the Affiliated Student Societies and through our members at large. Think of what it will mean to industry to have a thousand or more men each year entering upon their work with this splendid spirit, and with the ability to inspire confidence, good will, and loyalty. What will this mean to the industrial leadership of the future?

Discussion

Fred H. Rindge, Jr.,* New York, N. Y.—It is interesting to note that although this movement was started only seven years ago at Yale, it has spread in that brief time to 200 colleges and technical schools, and

* Traveling Secretary, Industrial Service Movement.

thousands of men are interested in it. There are 3,000 graduates who became acquainted with the movement in college days and are now promoting the industrial betterment idea all over the country. Some of these men are employed entirely in this work, and the number of such workers is constantly increasing.

We feel we are really doing a very fundamental thing in getting so many coming engineers and prospective employers of labor to interest themselves in these things which we have been discussing this morning; these things which are so vital to efficiency and success in any line of business or engineering to-day.

Of course the institutions located in our large industrial centers have the greatest opportunity for practical service. But I have just returned from one college, Pennsylvania State College, where 300 engineering students in three days signed up as interested in this movement and as anxious to take active part in the work this summer. That is one of a number of colleges situated in towns where there are no industrial opportunities. Therefore, their field of service must be where they work or live in the summer months or after graduation, and a large portion of those 300 men will be active this summer. There are men here who, if time permitted, could speak with conviction of the influence of this movement on their respective colleges.

There is another thing that should be especially noted, and that is indicated in Mr. Channing's paper, on the fifth page, where there is a rather specific illustration of the reaction on the student, the coming engineer, showing in just what way the experience helps him, and then the paragraph toward the close of the seventh page, and from there on through to the middle of the ninth page, further illustrates the other activities which the movement is promoting.

Mr. Channing did not take time to mention a number of the other practical features involved in this work. Possibly one of the most significant is that at many of the colleges and technical schools there is given a series of special lectures on the human side of engineering, accident prevention, and other things, which these coming employers must be interested in. One significant fact is that a number of colleges are planning to put in the curriculum required courses on the human side of engineering. That is a very important step and one which will be especially welcomed by practical business men and the alumni who now feel so greatly the lack of such instruction in their own training. When we have in our engineering schools a course on the humanics as well as the mechanics, it will be a big advantage to industry. It is surprising how the already full curriculums of the colleges can be readapted to include such important instruction as has been indicated.

It is clear that the movement is striking at some fundamental problems of the day and is helping constructively in many ways. Additional information regarding the whole plan may be obtained from the publi-

cations of the Industrial Department, 124 East 28th St., New York, which will be sent on request.

S. A. Taylor, Pittsburgh, Pa.—A little experience we are having in Pittsburgh may be of interest. About three years ago I took up with the representatives of the School of Mines of the University of Pittsburgh, the miners, and the Y. M. C. A., the question of working out a system of co-operation for the different mining camps around Pittsburgh. The result of that effort is interesting. We first tried to establish in the school a Saturday class for the men from the mines, in order to prepare them for the examinations they would have to pass for mine foremen or fire boss. The result was not a complete success. We found the men would not go into the school on Saturday, although a number of their employers were willing to pay their way and give them their day's wages. We then followed the saying of Mohammed, that if the mountain would not come to us, we would go to the mountain, and the result of it was that we established a series of night classes at the different mines in connection with the Y. M. C. A. work. The result was very satisfactory. During the year 1912–1913 we had 137 students in these night classes—I cannot give you the number of foreigners in those night classes, but there were a number. Some of the men had also previously taken some work in connection with the Y. M. C. A. classes. At the expiration of the first year 50 men out of the class of 137 tried the State examinations for mine foremen and bosses, and all but two passed.

The plan worked this way: The Y. M. C. A., through its organization, would arrange for the class, and the university would furnish the teachers. We had only two men we could spare for this work, and inasmuch as they would be obliged to move from place to place every day in order to take care of the night classes, we arranged with each of the classes to bear the expense of the instructor, simply his car fare and his hotel bill. The university paid the salary of the instructor.

A small, inexpensive hand lamp helped to make the system of teaching effective. By means of the reflector the illustrations, taken from advertisements in various technical journals, were thrown on the screen, and the machines and their operation explained. The instructor could carry this lamp with him from place to place. The result of that trial was a demand for this form of instruction so great that it was impossible for the university to carry it on, the funds being insufficient. I think the University of Illinois and the University of West Virginia are carrying on similar work.

There is another feature of this work which is worthy of mention. The manager of one of the companies to whom I proposed this matter said: "Yes, I will be glad to co-operate in this work, because I have carried out similar work in connection with the Y. M. C. A." He told me that by getting his men interested in this class of work he had

reduced a great number of the little agitations and complaints which came to him through the labor union. Men who would ordinarily go to the labor union meetings and talk and agitate matters, instead went to these night schools, and the result was that his troubles at the mine were greatly reduced.

In connection with that work we have established at the university a co-operative system, which all students taking the engineering course must enter to the extent, or the equivalent, of 10 weeks. There are only two weeks' vacation in the entire year. One-quarter of the entire year must be spent in the shop, in the engineering course, and in the case of the mining students, in or around some of the mines. We have found this an efficient way of bringing the students in close contact with the men in the various occupations. I believe it would be worthy of consideration by any of those who are so situated that they can avail themselves of such an opportunity.

H. H. Stoek, Urbana, Ill.—I think it is only right I should say a word, because Illinois was the first State in which such work was authorized to be done by the State, or through a State organization. Several years ago authorization was given to carry on the miners' and mechanics' institutes, but no appropriation was made, and only 12 months ago an appropriation of $15,000 became available for carrying on the work. Since that time 17 night schools have been established in Illinois, and about a thousand men are now receiving instruction in these schools. The work has been fully described in the *Bulletins* of the Miners' and Mechanics' Institute. There is a decided demand for such work and we find in Illinois that its scope and extent are limited simply by the amount of money available. We have asked the Legislature to give us $52,000 next year to carry on the work, and if that amount is granted, we can put a night school in every important mining center in the State of Illinois. The difference between what we are doing now and what can be done depends on the appropriation. We have had a number of inquiries from mining companies wanting young men trained for night-school and similar extension work.

Elmer L. Corthell,* New York, N. Y. (communication to the Secretary†).—This paper has interested me greatly and I fully believe in the excellent work outlined, and I may say I am in a small way assisting in it.

I know of the good work being done along this and other lines by the Y. M. C. A., particularly in other countries. I was one of the founders of the branch association in Buenos Aires in 1901, where now there is a membership of over a thousand and where the branch owns a very fine building for its various purposes. Again, I was in Switzerland at the time Dr. John Mott undertook to form several branches of the Inter-

* Civil and Consulting Engineer. † Received Mar. 22, 1915.

national Christian Students' Union. I assisted financially in establishing a branch at Berne, and I am still one of its annual supporters.

The reflex action upon the students themselves in the Industrial Service Movement which Mr. Channing describes is a very great benefit to them. It could not be otherwise, for it leads them to be useful men and to get into the habit of *service and self sacrifice,* and this lays a foundation for not simply *making a living,* but for *living a life.*

I think any engineering organization like yours could properly support such a movement and appoint a committee on "Industrial Service."

PHILIP W. HENRY, New York, N. Y. (communication to the Secretary*).—Mr. Channing's paper is of great interest and should be read carefully by all those occupying executive positions and by those who expect to fill them, because it shows what an important part the knowledge of human nature plays in dealing with labor and kindred problems. Such knowledge cannot well be gained from books, but must be gathered at first hand, by direct personal contact. Fortunate is the young engineer who—perhaps by force of circumstance—begins his career in a position where he associates on equal terms with the laboring men—using this term in its broadest sense. The insight which he thus gains into the real feelings and thoughts of the men who compose the membership of labor unions will be of great value to him later on when, by virtue of his technical education or superior natural talent and industry, he is occupying a position of responsibility, where his success may depend largely upon his ability to secure the hearty co-operation of his employees.

As most engineers are denied this intimate personal contact with the laboring man, the opportunity afforded by the Industrial Movement, as outlined in Mr. Channing's paper, should be heartily welcomed by the student engineer; for although primarily organized in behalf of the laborer, with the student as instructor, the teacher, rather than the pupil, is the one who obtains the greater benefit.

As engineers are coming more and more to fill executive positions, the knowledge of human nature becomes more and more important. Unless an engineer is able to select and place the proper men for the positions under his charge—a round man for a round position, and a square man for a square position—he will never hold for any length of time an important executive office. In other words, he must be an engineer of men as well as an engineer of materials. It is just as serious a mistake to select for mine manager a man who has only the qualifications for salesmanship—and *vice versa*—as it would be to select iron wire instead of copper for a transmission line.

In the past little emphasis has been placed upon the importance of the human element in engineering operations, and the Institute is to be congratulated on having a paper on this subject presented by a man of such broad experience and high standing as Mr. Channing.

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Safeguarding the Use of Mining Machinery

BY FRANK H. KNEELAND, NEW YORK, N. Y.

(New York Meeting, February, 1915)

SAFETY FIRST is a popular motto—most mining companies have adopted it. It is probable, however, that in the majority of cases it is only a motto and gets no further than the office stationery or the bulletin board at the mine's entrance.

In but few industries is there employed a greater diversity of machinery and mechanical devices than in coal mining. The list ranges all the way from mechanical stokers to wood-working machinery and undercutters. Many of these devices, particularly those employed in wood working, are extremely dangerous.

In 1913, 23 per cent. of all fatalities occurring in coal mining in the U. S. were caused by machinery of some sort or other. Although the forms of accidents occurring with mining machinery are legion, they generally arise from one or more of five causes: (*a*) Falls from ladders, platforms, etc.; (*b*) coming in contact with moving machines or parts thereof; (*c*) electric shocks; (*d*) failure of some machine part; (*e*) mismanipulation of valves, levers, switches or other hand-operated controlling devices.

The steps which may be taken to prevent accidents from the above causes are almost as numerous as the accidents themselves. There is, and can be, no panacea for all mishaps, nor for any one class of accidents. There is no mathematical or other formula that will bring immunity under any set of conditions. Common sense is the only guide.

The remedy for the first-named class of accidents is simple and generally effective—namely, make the staging, ladder, platform, or other support, abundantly strong to carry any possible weight that it may be called upon to bear; provide ample railings on platforms or runways, and non-slipping feet on movable ladders.

Coming in contact with moving machine parts is one of the greatest dangers met with in the operation and maintenance of mining machinery. Unfortunately, machine manufacturers have not as a class adopted the idea of thoroughly protecting and incasing the dangerous parts of their product. No one has, or should have, a better or more thorough knowledge of what the dangerous parts or elements of a machine may be than its builder. The desire to construct a piece of mechanism as cheaply as

possible impels many manufacturers to forego making the machines as safe as possible.

An exposed train of gears, an unprotected revolving bolt head, setscrew, feather or spline, an open keyway on a shaft, and the like, are all well-recognized dangers. Yet few indeed are the machines where these are all absent. The means to be employed to prevent accidents from these causes are two in number: First, eliminate so far as possible angular projections on revolving or moving parts; *e.g.*, replace ordinary square-head setscrews with those of the hollow variety which screw down flush. Second, guard or protect the moving parts of the machine.

When purchasing new equipment many firms specify something like the following: All angular moving parts or projections must be avoided as far as possible. If impossible to avoid their use, all gears, cams, eccentrics, setscrews, bolts, sprockets, chains, belts, feathers, splines, keyways, cranks, connecting rods, or other dangerous revolving or moving parts actuated by power, must be so guarded and protected as to render contact, intentional or otherwise, between them and the anatomy or clothing of the attendant as nearly impossible as the work to be performed will permit. All guards must be so constructed and built up as to be readily removable without the use of special tools, allowing ready access to the working parts for inspection or repair. It must be the aim of the builder to produce a machine that shall be as safe to handle and operate as human ingenuity can devise and the work in hand will permit, and no bid will be considered unless detailed plans are submitted which shall meet the entire approval of our engineers and safety experts.

But many machines are at present in use which were not purchased on the above or similar specifications and which represent too great an investment to be discarded. To render these machines even reasonably safe, adequate guards are necessary. These guards may be of many descriptions, but in general their object is to render an accident impossible. This is, however, out of the question, since the human element is an unknown quantity, and while it may be possible to make a machine foolproof, it is difficult to render it impregnable against suicidal intentions.

Several different materials may be employed with fairly good success. Wooden fences or boxes are in general the cheapest in first cost, but are obviously open to many objections. Hand railings and the like made of small structural shapes are, generally speaking, equally efficient and far more permanent than their wooden counterparts. Wire mesh of suitable weight, properly stiffened with structural-shape frames, so constructed as to be easily separated into sections, is one of the most efficient and permanent of all guards. Recently, expanded metal has been introduced in lieu of the wire mesh and appears to be admirably adapted to guarding purposes.

Generally speaking, an air gap of sufficient width is a certain protection

against dangerous electric shocks. While insulation gives supposed immunity, the only safe method of procedure is to treat every electrical conductor, regardless of covering or supposed voltage, with the same respect that would be given were it known to carry a large amperage at high potential. Although underground circuits at American mines usually carry direct current of a voltage not exceeding 500, yet we not infrequently hear of men being either injured or killed through forming a short circuit between trolley and rail. Trolley wires may be protected from accidental contact by comparatively simple means. The trolley guard board, and its means of suspension, are well known and need no further comment. Such protection should assuredly be placed at points of danger, particularly where men pass under the wire, and at other places where observation would indicate that accidents might occur.

A point of danger which exists at many mine plants, and one which is perhaps not generally recognized, is the space behind switchboards, particularly those carrying high-tension conductors or making high-tension connections. Such places, regardless of whether the current carried is direct or alternating, or whether the voltage is 110 or 110,000, should be effectually fenced in, and no one should be allowed behind the switchboard while current is on.

The failure of machine parts is the least probable of all accidents, but one of the most difficult to forestall. It is practically impossible to determine the existence of a faulty weld in a steam pipe, or a cold shut in a casing. Both may withstand a test pressure yet fail in service through the fatigue of materials.

It is well in designing a plant to provide good means of egress from the vicinity of all pipes carrying a high pressure of steam, air, or hot water, since the greatest danger from their failure is not so much from flying pieces as from the fluid released. In certain instances, breaking machine parts, dangerous in themselves, may be rendered harmless by proper means. The best known of such devices, although not the only ones, are gauge-glass protectors on boilers and safety collars on emery wheels.

Mismanipulation of hand-operated controlling devices is a prolific cause of accident, of which the overwind is perhaps the best known example in mining experience. Another type of apparatus which causes many deaths annually is the electric switch, which if thrown while a man is making repairs to the line it controls introduces an extreme hazard to the repairman. Boiler washers while at work are sometimes killed through the opening of some valve connecting to another boiler or to the feed pump.

There are upon the market many types of reliable speed regulators, automatic stops, and devices to prevent overwinding which may be attached to any hoisting engine, either steam or electric. A switch controlling a circuit should be provided with means for locking it open, and

the person making the repairs should see that the switch is so secured and have the key in his pocket. In the case of the ordinary single or double knife switch a hole properly placed in the switch blade and a padlock answer every purpose. So far as valves are concerned, the wheels may either be chained and locked, or a cover placed over the valve wheel, which when padlocked in place renders it impossible to move the hand wheel.

The above are a few of the preventive means of forestalling accident; the list might be greatly extended.

It is an old and a true adage that "A word to the wise is sufficient." Placards and danger signs, if properly distributed, and, what is vastly more important, if properly heeded, are invaluable. Such signs, however, should be of a character to be universally understood and therefore should be, so far as possible, pictorial or symbolic; that is to say, wordless. The clenched hand grasping the thunder bolts comes as near to conveying a definite idea to all who see it as does the wooden Indian in front of a cigar store. The red circle or disk has its significance also and needs no explanation.

The mental attitude of the employer and his lieutenants is a powerful factor in accident prevention. The superintendent, the foreman, and the pit boss are logical leaders to whom subordinates naturally, although perhaps unconsciously, look for example. If they are reckless, recklessness is apt to prevail among the men. If they are eternally cautious and vigilant, caution and vigilance will be inculcated. It does little or no actual good for a shop superintendent or foreman, for instance, to tack up a danger sign beside a motor and then send a man up a ladder to oil a running lineshaft upon which there are projecting setscrews, or keys, or boltheads. The foreman's example is vastly more powerful than his admonition.

In no branch of mine work should discipline be more strict, than in the enforcement of safety regulations. Breaches of ordinary rules mean little more than slight added expense to the mine management. Disobedience of safety precautions endangers not only the mine, but every living creature therein. The distinction between the two should be as clearly defined as that between civil and criminal actions in law.

The attitude of the mine workers throughout the country varies widely. In Arkansas some time ago the miners at a plant went on strike because certain safety statutes were not complied with. The employees of one of the largest anthracite companies of Pennsylvania strongly protested against, and even threatened to strike because of, the inauguration of what they called the "patrol" system, wherein a sub-foreman visits each working place at frequent intervals to see that all is safe and in accordance with the rules of the company. In many instances, the miner and mine laborer at least have failed to grasp thoroughly the idea that safety regulations are made and enforced for their

sole benefit, and that the necessity for the sub-foreman's instruction to set a prop or take down a dangerous piece of slate is a direct reflection upon their ability as efficient and careful miners. The good and prudent coal digger never fears the visit of the sub-foreman any more than the honest banker dreads the periodic call of the State bank inspector.

Among men who handle mining machinery there is far less antipathy to safety rules and regulations. The mental attitude of these men as a class is much more favorable to greater caution and security than that of the miners. The result is that they are much less prone to object to even decidedly stringent measures and frequently become safety enthusiasts.

Much effort, time, and money have been spent throughout the country during the past few years in first-aid work. This, although by no means a preventive measure, has perhaps exerted a far more powerful mental influence than all others put together. The first-aid field day, the working out of problems involving one or more imaginary, yet possible, injuries, the continual binding up of dummy fractures, etc., cannot fail to make a decided impression upon all who see them, particularly the participants. What we need in mine safety is prevention, not cure, and first-aid is not primarily preventive.

Superstitions and deep-rooted ideas are alike tenacious and require both time and effort for their effacement. We have long been accustomed to regard the calling of the coal miner as being inherently dangerous and requiring a certain amount of recklessness and bravado. The man who would not, on occasion at least, take a chance was not a good miner. And this idea has prevailed since mining began.

The movement for greater security to life and limb, therefore, is one which has to combat, not only existing physical conditions, but all the precedent, precept, and prejudice of the past. It is no wonder, then, that progress is slow, and that it often appears, even to the most sanguine, that a point has or soon will be reached beyond which material headway cannot reasonably be made. This is, however, only a feeling common to all movements toward betterment. Many of us can well remember when an 80-ton locomotive was considered as being about the largest that would ever be commercially successful. Just as the difficulties in the way of heavier rolling stock on the railroads lay not so much in the materials employed as in the preconceived ideas of the builders, so the barriers to greater safety in mining lie in the mental attitude of those concerned rather than in existing conditions.

It is futile to cover a machine with guards and safety appliances and depend upon them alone for protection. These are in the true sense effective only when supported and reinforced by discipline and morale. No guard ever devised is adequate to protect the man who is always careless. Habit is stronger than nature, and in mining no man is safe unless he is safe habitually.

Discussion

B. F. Tillson, Franklin Furnace, N. J.—I think it may be of interest in respect to the information we have received in regard to the danger of machinery in mining operations, to state that in the particular type of metal mining which the New Jersey Zinc Co. is doing at Franklin, during the past year, although there have been other accidents, that method of mining was free from any accident due to machinery underground. It may give a different light as to the relative degree of the importance of machinery as varying with different types of mining.

Arthur Williams, New York, N. Y.—Mr. Kneeland's able presentation of this paper, it seems to me, bears out the suggestion which has been so frequently made that in this country we have a new profession, that of the safety expert or safety engineer, which calling, if properly carried out, would include also questions of sanitation. While Mr. Kneeland was speaking, it occurred to me that perhaps any one, however experienced he might be in mining generally, would pass through these mine shafts and these operating rooms, and the things which might cause accidents, which would cause accidents if the right conditions were to occur, would pass by entirely unnoticed by him, but when the safety expert goes into these places he sees these things which he immediately recognizes as being spots of danger and provides means for their elimination.

It seems to me the time is coming when the moral, if not the legal, responsibility resting upon any employer or controller or contractor of labor, for any accident resulting from the permitting of these danger spots to exist in his work, cannot be evaded. I say, at least, morally, but I think the time is coming when it will be made a criminal act for any employer or contractor to permit danger spots to exist where human beings are engaged in industry. I think the time is not far distant when we will look back with a certain degree of, I will not say horror, but a certain amount of wonder, why for so many years in all industries we have permitted these danger spots to exist. They are not confined to mining; they are to be found everywhere, in every industry. The conditions surrounding the flywheel on that high-speed engine, shown by Mr. Kneeland, will be duplicated, I think, something like 500 or 1,000 times in large buildings in this city. The conditions where men can put their hands in gearings and be caught in rotating machinery will probably be found to be duplicated in very many of the large buildings in this city and other cities. A single manufacturer by sending safety experts through his works has reported that he was able in one year to eliminate something like 8,000 danger spots which were, at least, potential possibilities of serious accident to the human beings entrusted to his care.

Of course, we may think at first of the humanitarian side of this thing, but it goes beyond that. It seems to me there is an absolute definite responsibility resting upon any employer of labor to make the conditions surrounding the work conducted under his direction as safe as human experience can make it under the conditions existing at the time. But there is another side to it, and that is the safety, the higher efficiency of the working force, the better understanding between capital and labor, the advantage of leading the men to think and realize that the employers are not indifferent to their lives, health, and general welfare, and then there is the direct saving which comes from the elimination of accidents, in the larger amount of money that can be retained in the treasury of the corporation or employer. The Committee of Award for the Anthony N. Brady Memorial Medals, found and stated in their report that as a result of safety efforts in the case of one of the large electric street railway companies the expenditure for accidents, and on account of claims arising from accidents, had been reduced from 8.65 cents out of each dollar of gross income—8.65 per cent. of gross income—to 4.15 per cent.; in other words, the safety efforts of this corporation, with all that it meant in reduced accidents and better feeling toward the corporation, had actually resulted in saving about 4.5 per cent. annually upon the corporation's gross income, which might be more than 10 per cent., perhaps 10 to 15 per cent., annually saved upon the corporation's net income.

I believe that in the mining industry, or any other industry, where conditions are made safe, where pure air is supplied, where there is an abundance of good light, where machinery is safeguarded so that the men cannot slip into it, whatever their condition may be, the higher efficiency, added to the saving of the cost of accidents, may mean a large increase in the net returns upon the employer's operations.

Housing and Sanitation at Mineville

BY S. LEFEVRE, MINEVILLE, N. Y.

(New York Meeting, February, 1915)

THE solution of the housing and sanitation problem in mining communities, keeping in view both economic and humanitarian aspects, demands the best thought of the management of such enterprises. Upon the solution depend the health and comfort of the employee and his family, and as a consequence his contentment and efficiency.

The following description covers briefly the methods installed for Witherbee, Sherman & Co. The operation of the system has been for about two years under the direct supervision of T. A. Hammond, Land Agent, and I am indebted to him for a compilation of the results.

Sanitation

The sanitary service covers 238 tenements. The installation cost was about $22 per tenement, as follows: Privy, $15; privy box, $3; proportion of incinerator cost, $2; garbage can, $2. The operating cost is $1 per month for each tenement.

In considering the installation of an ordinary village system of water supply and sewerage two difficulties were encountered:

First, lack of a sufficient quantity of water; there is no stream large enough, as Mineville is situated very close to the divide between the Lake Champlain and Hudson River watersheds. A storage reservoir was tried, but the only location available, among glacial hills, would not hold water.

Second, the expense of digging ditches for the distribution of water and sewer pipes is so great as to be prohibitive; Mineville is built chiefly in a valley covered in the bottom with drift, in this case a combination of boulders and hard pan. Digging in this material costs 80c. to $1 per cubic yard, and pipes must be placed 4½ ft. down to be below frost. The houses are scattered along 4 miles of streets, the oldest having been built in the original forest 65 years ago. The newer houses have gone up a few at a time wherever a clear space could be found and the slopes were not too steep. This has advantages, as it separates the dwellings into

various groups, which makes it possible to segregate the different nationalities; thus we have an American quarter, an Italian, and a Polish-Slavish-Hungarian district.

A water pipe leads to the works for boiler purposes and this supplies the houses along its line. One group of six obtains its water from a well on the hill above; a windmill pumps into an adjacent concrete tank. In

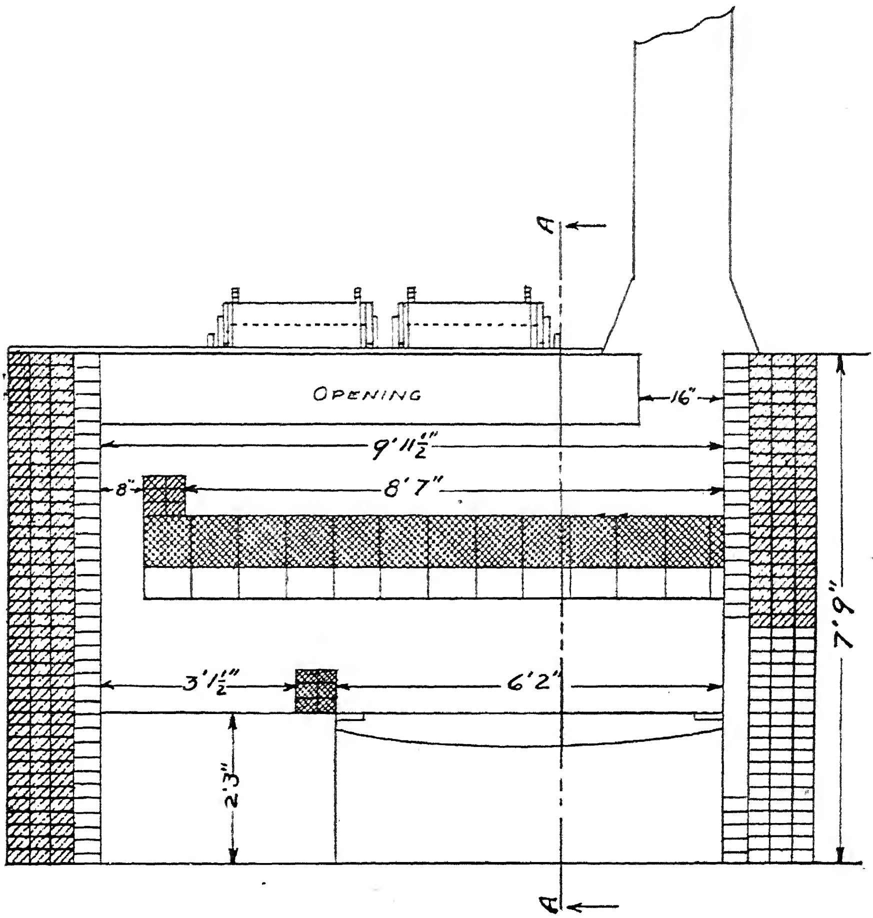


FIG. 1.—LONGITUDINAL SECTION OF INCINERATOR.

several other cases an electric pump in the cellar delivers the water from a well into an air-pressure tank, also in the cellar. Sixty per cent. of the houses are supplied from wells. To keep them free from contamination the old vaults have been abolished, and each privy supplied with a 16-gauge galvanized-steel box, size 14 by 14 by 30 in. Three hundred of these boxes were made to our specifications by the Lyon Metallic Manufacturing Co. They are soldered and riveted water-tight, have

one drop-handle on each end, and cost $2.73 delivered. With them were supplied 24 covers with handles and with rims 2 in. deep, the price being $1.25 each. The covers are kept on the boxes only while in transit from privy to incinerator. They are fitted with a strip of rubber packing on the inside next to the rim, and clamps are used to press the cover against the upper edge of the box, preventing the contents from slopping out. After the boxes have been emptied into the incinerator they are taken to a washing rack, thoroughly cleaned and set aside to dry. Before going into service again each box has a small quantity of lime placed in it. The small privies are built with one seat, the larger ones having partitions

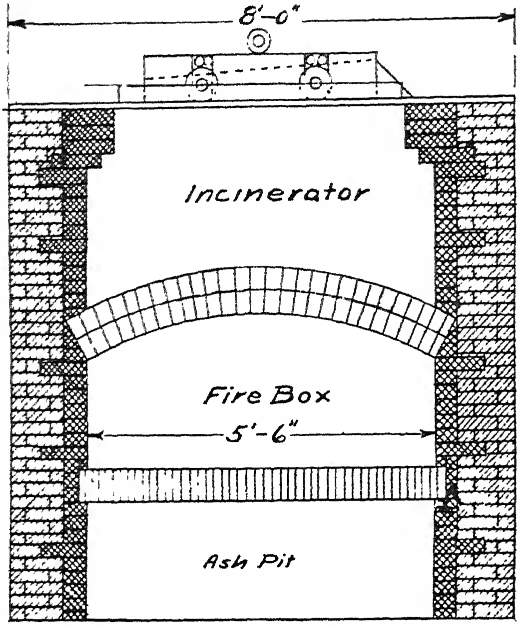


Fig. 2.—Secton on Line A-A of Fig. 1.

built between seats. The average cost is $15 set up. They are furnished with door and window, the latter being kept screened. The metal box is placed close up under the bottom of the seat, sliding in from the rear or end through an opening which is kept closed by a tight-fitting door. The boxes are removed frequently and are not allowed to become full. By this system we have been very successful in preventing the contamination of wells and springs.

A garbage can is provided at each house. These are emptied once a week, or oftener if necessary. They are No. 1 Witt cans with tight-fitting lids, made by the Witt Cornice Co., size 15¾ by 25 in., capacity 20 gal., weight 26 lb., made from one piece of 23-gauge, 2-in. corrugated galvanized steel, braced with heavy steel bands at top and bottom, and

cost $1.93 each at Mineville. They have kept their shape, stand handling and are satisfactory.

Incinerator.—An incinerator, shown in Figs. 1, 2, and 3, was built at a cost of $500, having a firebox of boiler grates 5 ft. 6 in. wide by 6 ft. 2 in. long. A brick arch over the firebox also makes the bottom of the incinerator chamber. An opening behind the bridge wall allows the flame to go above the material to be consumed and up the stack at the fire-door end. This consumes all the gases. No smoke is seen after the fire is started and there is not much odor. The location is about one-eighth mile from the nearest houses.

It was thought when built that the capacity would be large enough

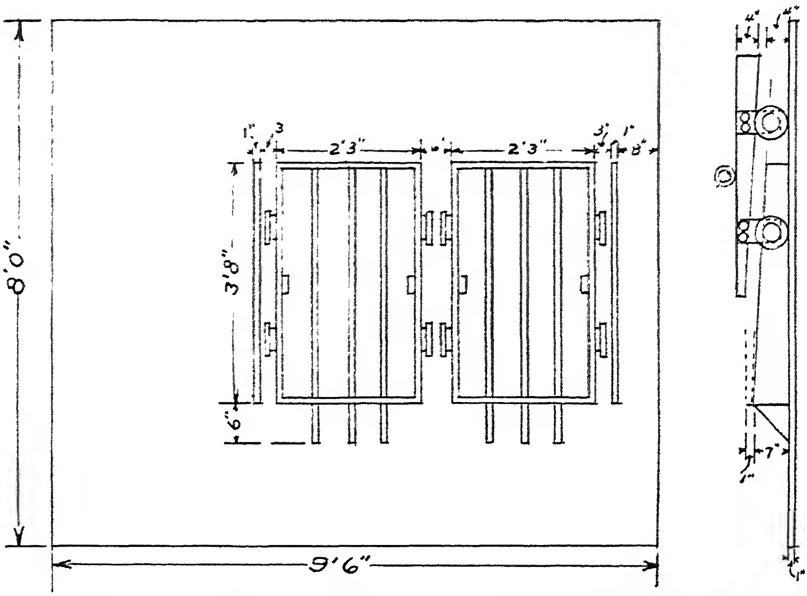


Fig. 3.—Top View of Incinerator, showing Construction of Tight-Fitting Doors.

to burn the material from garbage cans also, but it has been found that the incinerator will only take care of the privy boxes. A charge is put in each morning, after which the fire is started and kept going all day, being allowed to go out at night. In the morning the incinerator is cold enough to permit the cleaning out of the ashes, when needed, or for another charge. The top is cast iron, with two feed openings 3 ft. 8 in. by 2 ft. 3 in., closed by cast-iron doors on wheels, having a sloping bottom which comes in contact with a similar slope on the sides of the feed opening in the iron top. This contact makes a tight joint, yet a slight lift with a bar frees the door and it rolls open easily on the wheels.

The material from the garbage cans is now burned in heaps on the ground in the open air and is disposed of in this way without any trouble.

When the incinerator is rebuilt a double grate will probably be provided. The garbage, kept from packing too tightly for the gases to pass through by baffle rods, would be placed on the upper grate, where it would dry and afterward burn. In this way some fuel value would be secured from the garbage.

The cost of operating for the first six months of 1914 was: Labor, $1,200.35; fuel and lime, $213.75; total, $1,414.10, or $235 per month, about $1 per month per tenement. The fuel used during this period was soft coal, but generally enough refuse lumber from the saw mill, or discarded concrete forms or similar waste is available to answer the purpose. This was largely so in 1913, and no coal has been burned since July, 1914.

Two one-horse wagons are used for collecting the refuse. They are low slung to make loading easier. One man drives each wagon, but the rigs travel together when collections from privies are made, as the loaded boxes are too heavy for one man to handle. A third man fires the incinerator and keeps the yard in proper shape, burning the garbage in heaps. These three men with the two horses comprise the operating force.

A number of concrete septic cesspools have been built, each taking the waste from one or two tenements; the water discharged is clear, with no odor. The simplest form has no partition. The inlet pipe extends to within 2 ft. of the bottom, and the outlet pipe turns down inside about 1 ft. below the level of discharge. This keeps the scum on the water undisturbed and allows the microbes to do their work. If there is any odor from the discharged water it usually indicates that too great a volume of water is going through, possibly due to a leak in the water system.

A welfare worker is employed, who is a trained nurse and is connected with the hospital service. She visits the houses and reports, on a card furnished her for this purpose, any unsanitary conditions or cases of illness. One of her most important duties is to give suggestions as to the care and feeding of infants and small children. This line of work is very helpful and does the greatest good.

When the results of this work get as far as the schools, the company is in about the same position as the old woman who lived in the shoe. We have to provide an additional school-room and teacher about every year. There are 542 children of school age, and nearly as many more younger. The present number of men working is 600.

Housing

The total number of houses occupied is 238. Of these, 90 take water from mains, seven from wells with individual electric pumps, and 141 from wells or cisterns. There are 21 houses with toilets and baths, and 217 with privies.

The older wooden houses are mostly of the double tenement type, with four or five rooms for each family. During the past eight years no wooden houses have been built. Instead, concrete blocks made from the tailings of the concentrating plants have been used. The interior floors and partitions are wood, and the roofs of slate. The concrete blocks are 8 by 10 by 20 in.; each occupies 1$\frac{1}{10}$ cu. ft. of wall space. It costs 15c. to make a block, mixed 1 part cement and 6 parts tailings, this cost including sprinkling and piling for seasoning; 10c. additional covers the expense of hauling, handling, and laying in the wall, making the blocks cost 25c. each in the wall. No charge is made for the tailings, and the blocks are made at the foot of the tailings pile. The blocks are mixed

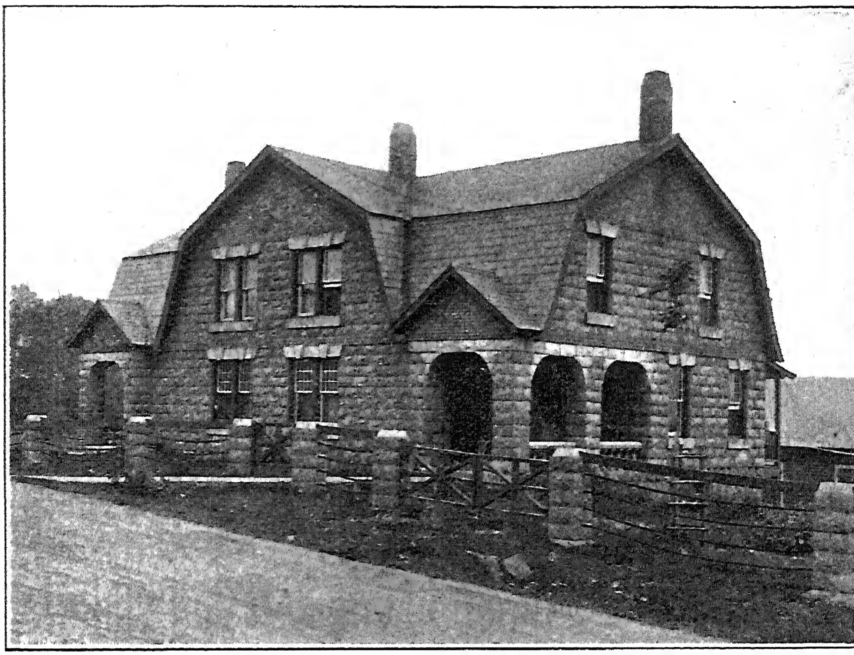


FIG. 4.—DOUBLE HOUSE OF CONCRETE BLOCKS AT MINEVILLE.

and tamped by hand, and made in a Century block machine. The 10-in. blocks have a 4-in. air space in the middle. On one of the first houses the experiment was tried of plastering the inside wall directly on to the concrete blocks, but this was a failure, as the water ran down the walls in streams; not due to moisture coming through the concrete-block wall, but to the condensation of moisture from the heated air inside coming into contact with the cold surface of the block wall. To remedy this trouble 1-in. strips were nailed to the blocks, to which wooden laths were nailed and the wall was replastered.

There are 88 concrete tenements of various types, single, double, and

four-family houses, including five larger boarding houses. The exterior and general arrangement of one of the double concrete houses are shown in Figs. 4 and 5. Most of those for American families are single houses with 50 to 60 ft. front to each lot, affording space for lawn and garden.

These concrete houses present a good appearance, are warm in winter, cool in summer, and are not damp. The cost of maintenance of the outside is practically nothing; inside it is the same as a wooden house.

The cost is about 10 per cent. more than wood. The secret of avoiding the sameness of appearance which spoils the effect of most concrete-block structures is in selecting the material to put in the face of the mold.

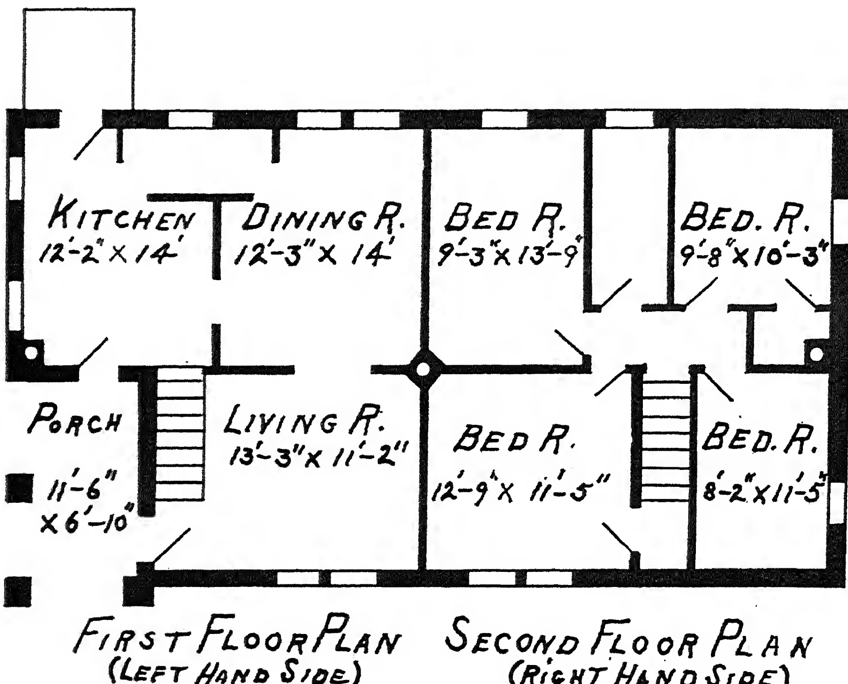
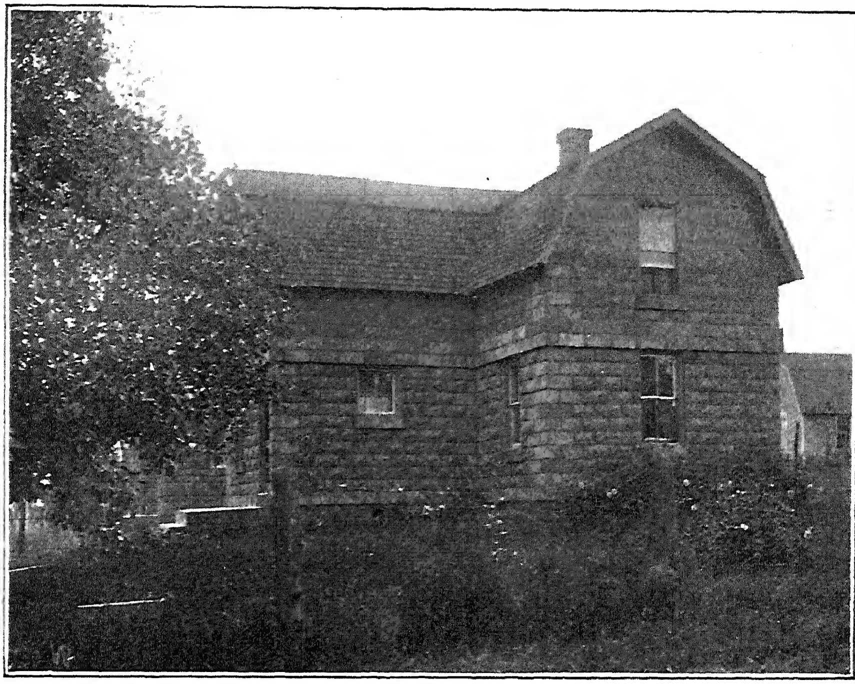


FIG. 5.—FLOOR PLANS OF HOUSE SHOWN IN FIG. 4.

If the face of one block is of moderately coarse material and the next one is all fine, when they are laid in the wall side by side the monotony is broken.

For the foreigners four-family tenements have proved quite satisfactory. These houses have a kitchen, a dining room and a family bedroom on the first floor, and two or three bedrooms on the second floor. Each family is a recruiting center, for when more men are wanted, they write their friends to come and get work and board with them. The larger boarding houses, accommodating 50 boarders, are not so popular, the men preferring to stay with the families who can take only five or six.



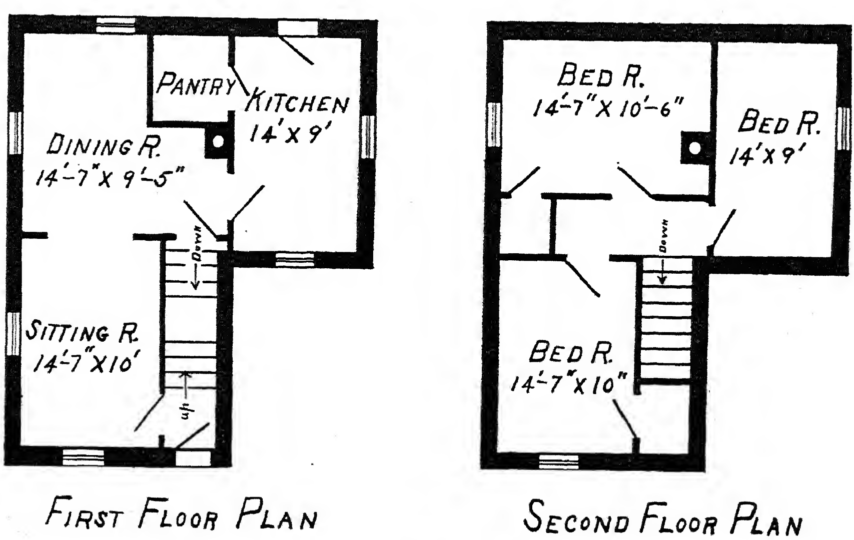


FIG. 6.—SINGLE HOUSE WITH SIX ROOMS; NO HEAT OR PLUMBING; SHINGLE ROOF. COST $950, AND INDIVIDUAL FRAME BARN, $100. RENT $8 PER MONTH, INCLUDING BARN.

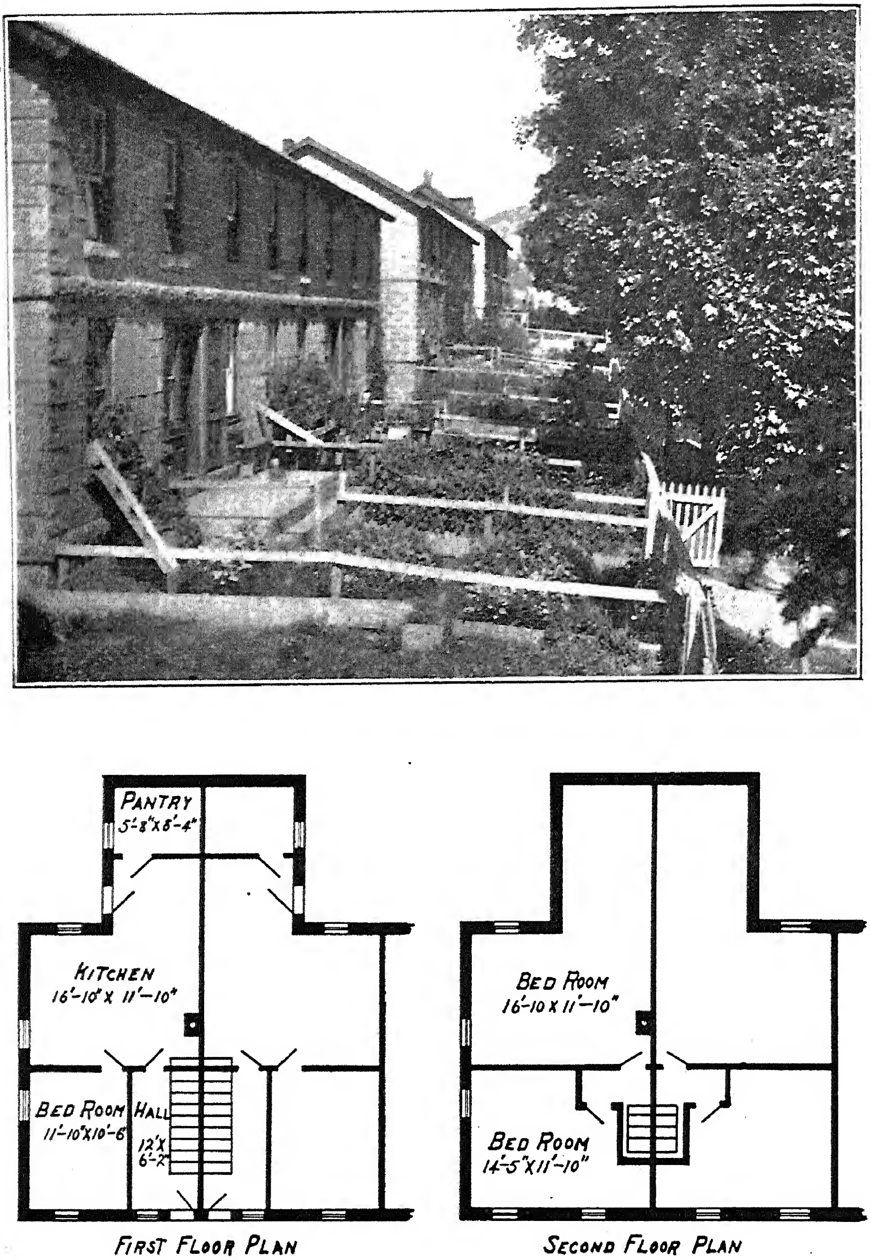


FIG. 7.—FOUR-FAMILY TENEMENT FOR FOREIGN LABORERS. SIZE 70 BY 26 FT.; CELLAR 70 BY 12 BY 7 FT.; SLATE ROOF. COST $3,000, OR 6C. PER CUBIC FOOT. RENT $5.50 PER MONTH, INCLUDING BARN.

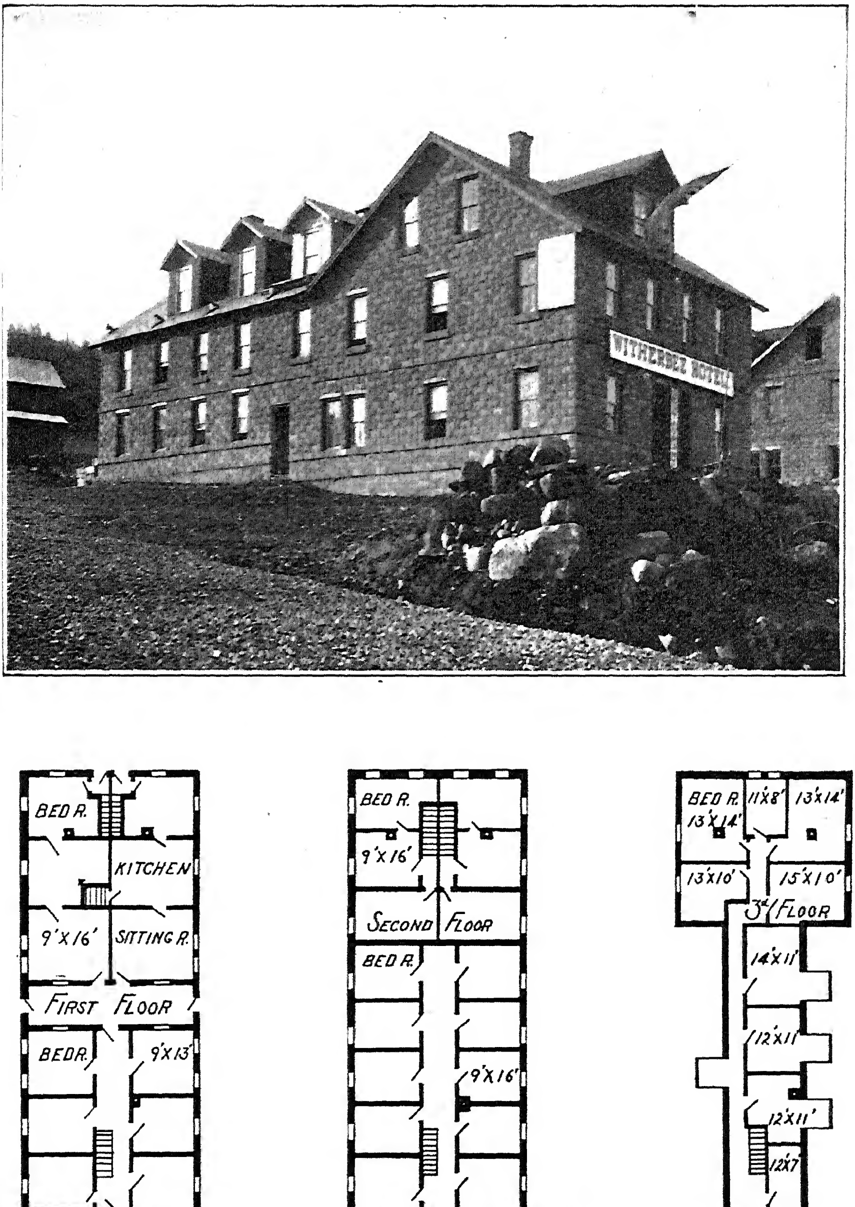


Fig. 8.—Boarding House to Accommodate 50 Men, and Two Families to Keep the House. Cost $4,000, without Heat or Plumbing. Rent $25 per Month for Whole House. In house shown in picture, steam heat and plumbing, laundry and wash room were installed at cost of $1,000 additional.

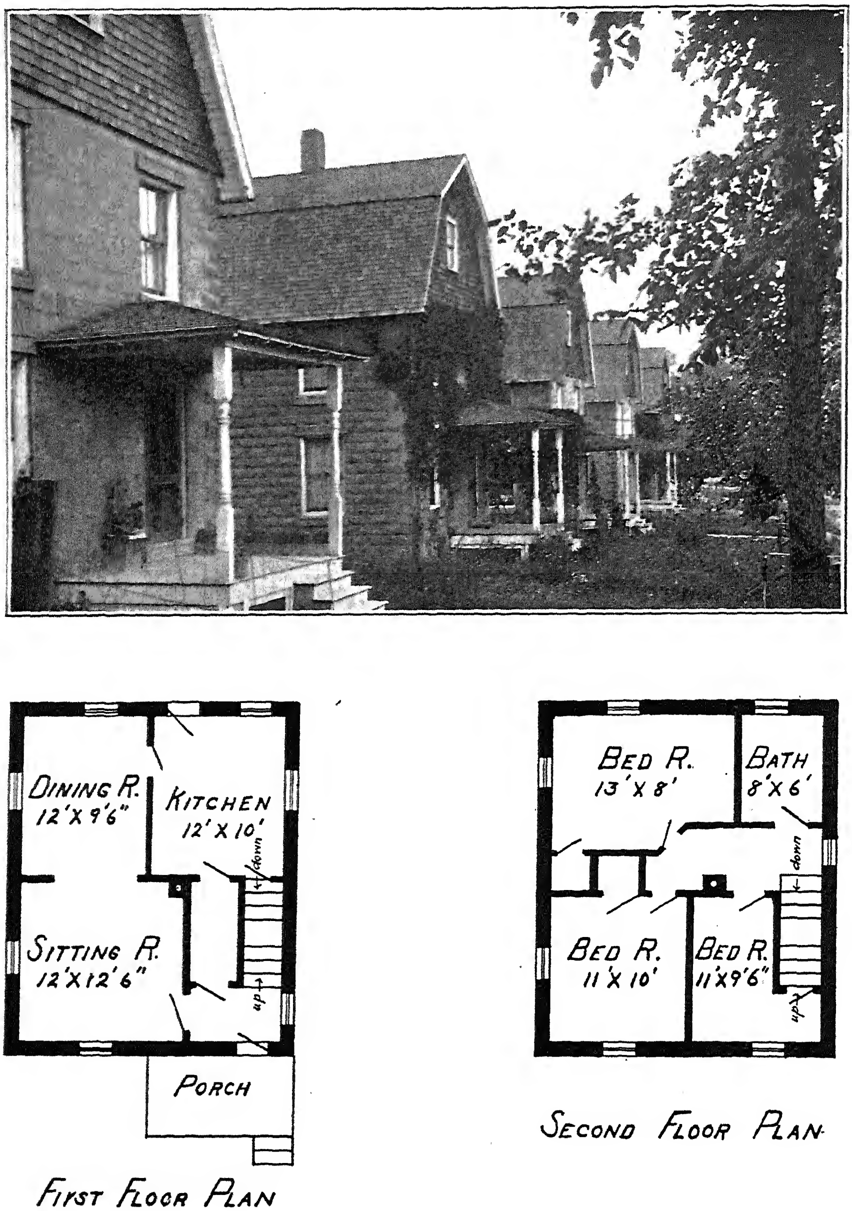


Fig. 9.—Row of Six Single Houses, Size 26 by 22 ft., Two Stories and Attic, Full-Sized Cemented Cellar. Bathroom, but no Heat; Water Supply from Windmill and Well. Cost $1,350, or 9 c. per Cubic Foot. Rent $9 per Month, including Barn. Each has a different combination of rock-faced and plain blocks.

Each tenement has a small flower garden for each family, and a space of 30 to 40 ft. is left between the buildings. Barns are found to be essential and two double barns are built for each four-family house, with accommodations for a cow, chickens, and a pig. The privies are built in a corner of the barns. This general arrangement avoids a nondescript collection of shelters in each back yard. Prizes given for the best-kept lawn, flower bed, and window box have stimulated interest and pride in appearances, and have added greatly to the attractiveness of the village.

The following regulations apply to the keeping of live-stock: Cows and horses must be kept in barns or pastures, and out of yards and off the streets. Pigs must be kept in barns or pens at all times. Fowls must be kept on owners' premises. Yards, barns and all buildings must be kept clean at all times. Any live-stock or fowls found roaming in yards or streets will be put in the pound and released only on payment of a fine. A herder is employed by the company

Experiments on the Flow of Sand and Water Through Spigots

BY R. H. RICHARDS, BOSTON, MASS., AND BOYD DUDLEY, JR., STATE COLLEGE, PA.

(New York Meeting, February, 1915)

In nearly all ore-dressing operations it is a common practice to discharge mixtures of fine ore and water through spigots; for example, from classifier pockets, from jig hutches, from settling tanks, etc. As a general rule it is desirable to regulate the composition, *i.e.*, the ratio of water to sand, of such discharged products; and this can usually be done by varying the size of the orifice or spigot through which the material is discharged. Thus it is that problems similar to the following are frequently encountered: With a given quantity of solids per unit time, of a given specific gravity, to be discharged with water under a given head, and with a given ratio of water to solid, what must be the size of the spigot opening? If the spigot is to be open continuously, there are, aside from its form, but three factors governing the rate at which it will discharge a mixture of sand and water, namely: the head of water above the spigot; the area of the spigot opening; and the viscosity of the material to be discharged. The term viscosity, as used here, means the ratio of the volume of pure water that will flow through a given orifice under a given head in a given time to the volume of the material under consideration that will flow through the same orifice in the same time under the same conditions. While the actual viscosity of water containing sand in suspension is not increased by the presence of the sand, nevertheless the volume-rate at which such a mixture will flow through an orifice is less than that at which pure water will flow, by reason of the friction of the particles of sand against each other and against the sides of the orifice. In other words, the flow is retarded as it would be by increased viscosity; and therefore, for the purpose in view, we may speak of the viscosity of a mixture of ore grains and water as defined above. The head of water above the spigot is generally fixed by the form and design of the classifier or tank to which the spigot is attached. So, with the head known, in order to determine the size of the spigot required it is necessary to know the viscosity of the desired mixture, and the coefficient of discharge of

the form of spigot employed when pure water is the material being discharged. From these data, the area of the spigot opening can be calculated from the equation,

$$a = \frac{fq}{c\sqrt{2gh}}$$

in which a is the area of the spigot opening; f, being the viscosity of the mixture; q, the rate of discharge by volume; c, the coefficient of discharge; g, the acceleration due to gravity; and h, the head of water above the spigot.

The coefficient of discharge is the ratio of the actual rate of discharge of pure water to the theoretical rate of discharge as calculated from

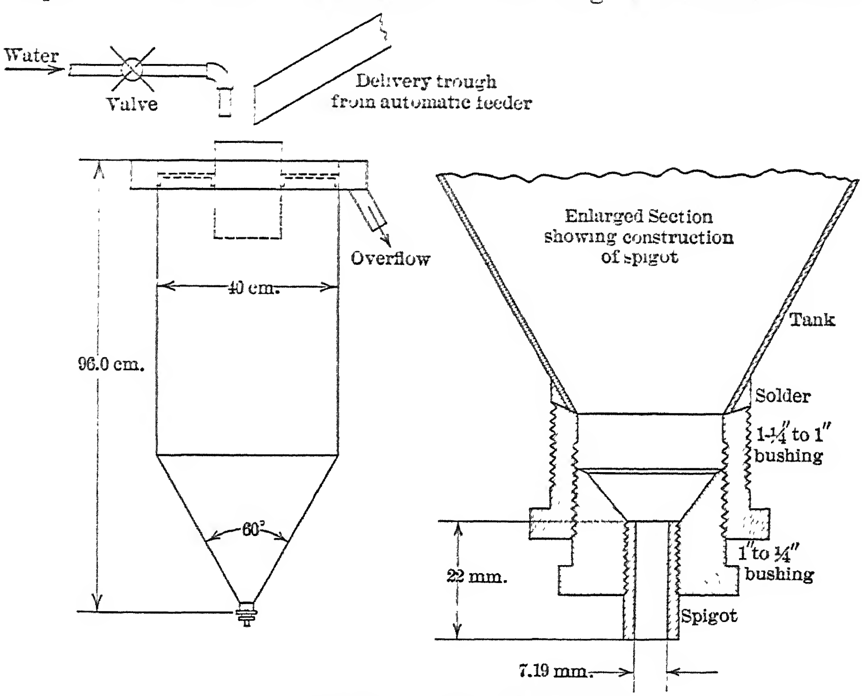


FIG. 1.—APPARATUS FOR TESTS OF FLOW.

the effective head above the orifice or spigot. The value of c depends upon the form of the orifice or spigot; some representative values taken from Merriam's *Treatise on Hydraulics* (8th ed.) are given below. For orifices having a sharp inner edge, which is alone touched by the water, c = 0.59 to 0.63, being greater for low than for high heads, greater for rectangles than for squares, and greater for squares than for circles. For a standard short tube (a cylinder having a length equal to about three times the diameter) not projecting into the space occupied by the water, c = 0.79 to 0.83, being greater for low heads than for high heads. For a standard short tube projecting into the space occupied by

the water, $c = 0.72$. For short tubes not projecting into the space occupied by the water and having a cone- or bell-shaped mouth on the influx end, $c = 0.85$ to 0.95. The latter form is the one usually found in classifier spigots, and a number of tests on the spigot shown in Fig. 1 under heads of from 1 to 3 ft. gave values for c between 0.87 and 0.91—the higher values for the lower heads.

The experiments described below were performed with the purpose of securing data as to the relation between the composition and the viscosity of mixtures of sand and water. The work was done during the winter of 1911–12 in the ore-dressing laboratory of the Massachusetts Institute of Technology.

The apparatus used in the tests consisted of a cylindrical tank of galvanized iron, having a conical bottom to which the spigot was attached, and a mechanical feeder for delivering dry sand to the tank. The details of the construction of the tank and spigot and of the arrangement of the apparatus are shown in Fig. 1. The tank was mounted with the top or overflow edge level; and, by feeding enough water into it to supply the spigot and to maintain a slight overflow, a constant head of water was maintained above the spigot. Below the tank a swinging sheet-iron launder furnished the means of deflecting the spigot discharge into a bucket or other receptacle for any desired time, and thus determining the rate of flow.

The method used in conducting a test was as follows: The tank was filled with water and the rate of inflow so adjusted as to supply the spigot and to maintain a slight overflow. The feeder was then started and sand was delivered to the tank. The sand settled through the water and passed out of the spigot. The amount of overflow was thereby increased, not only on account of the added volume of sand, but also because of the decreased rate of discharge due to the passage of the sand through the spigot. Nevertheless, this increase in the amount of overflow did not produce a measurable increase in the head above the spigot, because of the great length of the overflow edge and because the actual increase in the rate of overflow was comparatively slight. When the sand and water mixture had been running from the spigot for a minute or so and the flow had become steady, the spigot discharge was deflected into the bucket for a measured length of time, usually about 2 min. The feeder was then stopped, adjusted to deliver sand at a different rate, and another test was made. The discharged product obtained from each test was weighed to determine the total amount of water and sand. The water was then decanted from the sand and the latter was dried on a steam table, after which it was weighed.

The sand used in all the tests was prepared by crushing Roxbury pudding-stone (a siliceous conglomerate) with rolls to pass a wire-cloth screen with 1.4-mm. square holes, and removing the sand and slime finer

than about 0.1 mm. by hydraulic classification. The specific gravity of the prepared sand was 2.72.

The results of the experiment are shown in the following table. The rates of flow are all expressed in kilograms and liters per minute.

TABLE I.—*Relation of Composition to Viscosity of Mixtures of Sand and Water*

Kilograms Sand and Water	Kilograms Sand	Kilograms and Liters Water	Liters Sand	Liters Sand and Water	Per Cent. Sand by Volume	Per Cent. Sand by Weight	Viscosity of Mixture
9.20	0.00	9.20	0.000	9.20	0.00	0.00	1.00
9.30	0.45	8.85	0.165	9.02	1.83	4.84	1.02
9.35	1.10	8.25	0.405	8.66	4.68	11.8	1.06
9.35	1.40	7.95	0.515	8.47	6.08	15.0	1.09
9.40	1.90	7.50	0.699	8.20	8.53	20.2	1.12
9.40	1.95	7.45	0.717	8.17	8.78	20.8	1.13
9.55	2.20	7.35	0.809	8.16	9.92	22.0	1.13
9.20	2.25	6.95	0.827	7.78	10.6	24.4	1.18
9.05	2.50	6.55	0.920	7.47	12.3	27.6	1.23

The relationship between the composition and viscosity of the mixtures is shown graphically in Fig. 2. The ordinates of the upper curve are the percentages of the sand by weight, while those of the lower curve are the percentages of sand by volume. The viscosities of the mixtures are plotted as abscissas in each case.

It was found in these tests that when the amount of sand in the mixture exceeded 30 per cent. by weight the spigot would produce a very thick discharge for a short time but that its continuous operation under these conditions was not certain; clogging resulted sooner or later. It is therefore concluded that, with conditions similar to those under which these tests were made, a mixture of sand and water cannot be successfully discharged from a spigot when the mixture contains more than about 30 per cent. of sand by weight. In the case of the sand used in these experiments, this figure corresponds to about 13 per cent. of sand by volume, and it should be noted that it is the volume-percentage, rather than the weight-percentage, which governs the viscosity. With sand of greater density, a mixture having more than 30 per cent. of sand by weight could of course be discharged; but it seems doubtful whether a mixture containing more than 13 per cent. of sand by volume would pass continuously from this spigot if the sand grains were 1.4 mm. in diameter. On the other hand, with the same size of grain and a larger spigot or with smaller grains and the same spigot a more concentrated mixture could doubtless be discharged.

There is much more experimental work to be done along this line,

before a complete and satisfactory set of data will be available. Two points that call for investigation are the effect of grain size and the effect of the density of the grains upon the viscosity of the mixtures; another is the relationship between the ratio of the spigot diameter to that of the maximum grain and the mixture of maximum concentration that will flow.

A concrete example, illustrating the use of the data given above, may prove of interest. It is desired to discharge from the pocket of a classifier 40 tons of sand per 24 hr. together with water in the ratio of 1 part of sand to 3 parts of water, by weight. The head of water above the spigot

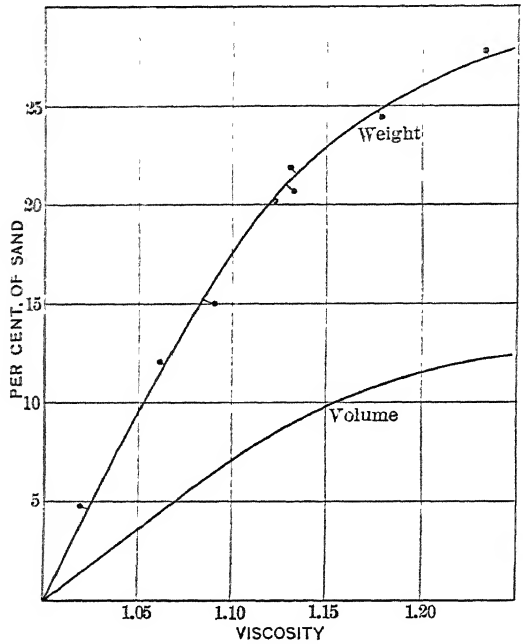


FIG. 2.—GRAPHIC REPRESENTATION OF RESULTS SHOWN IN TABLE I.

is 3 ft. The form of the spigot is that of a short tube with a cone mouth on the influx end. The mean specific gravity of the sand is 2.81. What must be the diameter of the spigot opening? For the sake of convenience metric units are used in making the calculation.

The area of the spigot opening may be obtained from the formula given above,

$$a = \frac{fq}{c\sqrt{2gh}}.$$

Taking up the terms on the right hand of the equation in order, f, the viscosity, may be estimated as follows: The weight-ratio of water to sand in the mixture to be discharged is 3 to 1. Considering 100 g. of the mixture, the weight of water is 75 g.; its volume is 75 cc. The volume of the sand is 25 g. ÷ 2.81 (the density of the sand) = 8.9 cc. The total

volume of 100 g. of the mixture is 75 + 8.9 = 83.9 cc. Hence, the percentage of sand by volume in the mixture is 8.9 ÷ 83.9 = 10.6. From the lower curve of Fig. 2, the viscosity of a mixture containing 10.6 per cent. of sand by volume is 1.17. Therefore $f = 1.17$.

The quantity of sand discharged per 24 hr. is 40 tons. One ton per 24 hr. is 0.631 kg. per minute. Forty tons per 24 hr. is 40 × 0.631 = 25.2 kg. per minute. The volume of sand per minute is 25.2 ÷ 2.81 (the density) = 8.98 liters. The quantity of water per minute is three times that of the sand, 25.2 × 3 = 75.6 kg. = 75.6 liters. The total volume of sand and water per minute is 8.98 (sand) plus 76.5 (water) = 85.5 liters. The total volume per second is 85.5 ÷ 60 = 1.43 liters = 1,430 cc.

Since the spigot is to consist of a short tube with a cone mouth on the influx end, the coefficient of discharge, c, may be assumed as 0.88.

In metric units $g = 980$ cm. per sec.^{2}; and the head, h (3 ft.), is 91.4 cm.

Substituting these values in the above equation gives, for the area of the spigot opening

$$a = \frac{1.17 \times 1430}{0.88\sqrt{2 \times 980 \times 9.14}} = 4.50 \text{ sq. cm.}$$

The diameter may be obtained from the relation,

$$d = 2\sqrt{\frac{a}{\pi}},\ d = 2.40 \text{ cm.} = 0.94 \text{ in.}$$

Discussion

R. H. Richards, Boston, Mass.—I believe that the manner of working for efficiency in almost all lines is to study the conditions of highest efficiency, and then to measure what is being done. We can then see how near we have approached the highest efficiency. Spigots in practice often use many times the water theoretically needed. There are places where that is necessary for safety reasons. There are, however, places where it is unnecessary, and in these places it seems to me that this work of Mr. Dudley's will be a great help to the mill man to arrive at the greatest economy of water and highest efficiency.

Albert R. Ledoux, New York, N. Y.—A good many years ago I received a telegram from that distinguished engineer, Hamilton Smith, which simply said in substance: "I have discovered that liquids will flow faster from a square orifice than from a round orifice of the same area, under the same head." I wondered why he sent me the telegram, but some time afterward he told me that he had been making experiments along that line in Montana; that he thought the discovery was interesting and that I would be pleased to have early information of it; and that he was going to make it public when his tests were completed.

For a year or two after that, I watched the technical journals to see if he had ever published anything on the subject, but I have never seen any printed statement along this line. A few years afterward—as we all know—Mr. Hamilton Smith died.

I mentioned this fact to learn if the authors' experiments verify Mr. Smith's statement; or if they have found a paper by Mr. Smith covering this matter, which I may have overlooked.

BOYD DUDLEY, JR.—Perhaps some of the information contained in this paper was originally due to Mr. Smith. I do not know. However, it comes from Merriam's *Treatise on Hydraulics*, and this is given—"For orifices having a sharp inner edge, which is alone touched by the water, C equals 0.59 to 0.63, being greater for low than for high heads, greater for rectangles than for squares, and greater for squares than for circles." That covers the point, I believe, that has been raised, but I do not know just to whom the credit is due.

Development of the Butchart Riffle System at Morenci

BY DAVID COLE, MORENCI, ARIZ.

(New York Meeting, February, 1915)

THE appearance of the Wilfley table in 1897 marked an epoch in the art of concentration of ores. The table has merited and received an almost unprecedented measure of public approval, lasting through its whole patent life. It has been very little improved in itself, or improved upon by competitors. The new machines bidding for popular favor have been of the Wilfley family, but without ability to do markedly better work than the parent machine, or to do more work except by the use of superimposed decks, the latter having obvious disadvantages.

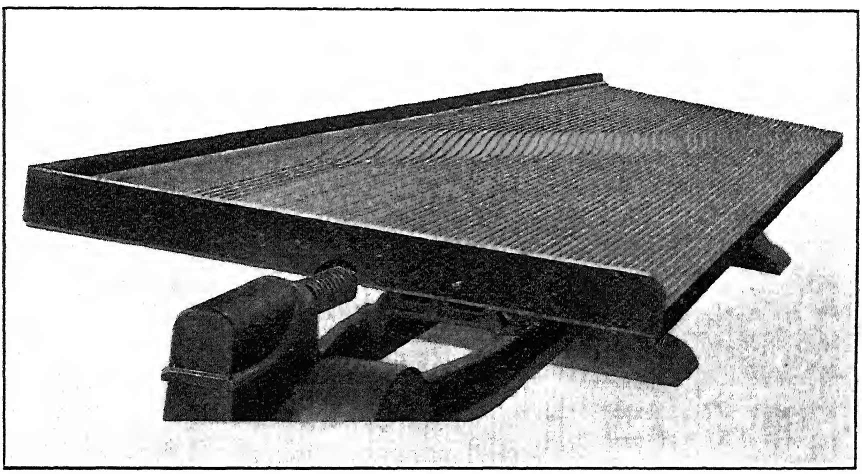


FIG. 1.—THE BUTCHART RIFFLE SYSTEM.

During the past three years there has been developed at Morenci a new type or arrangement of riffles applicable to the Wilfley type of concentrating table which corrects many of the objectionable features or limitations of the older system and obviates most of the difficulties encountered with it, particularly when handling ores having a low ratio of concentration. The new type is known as the Butchart[1] riffle system, and its general arrangement is shown in Figs. 1 and 2.

The old system accomplished stratification satisfactorily but has

[1] Also known as the National table.

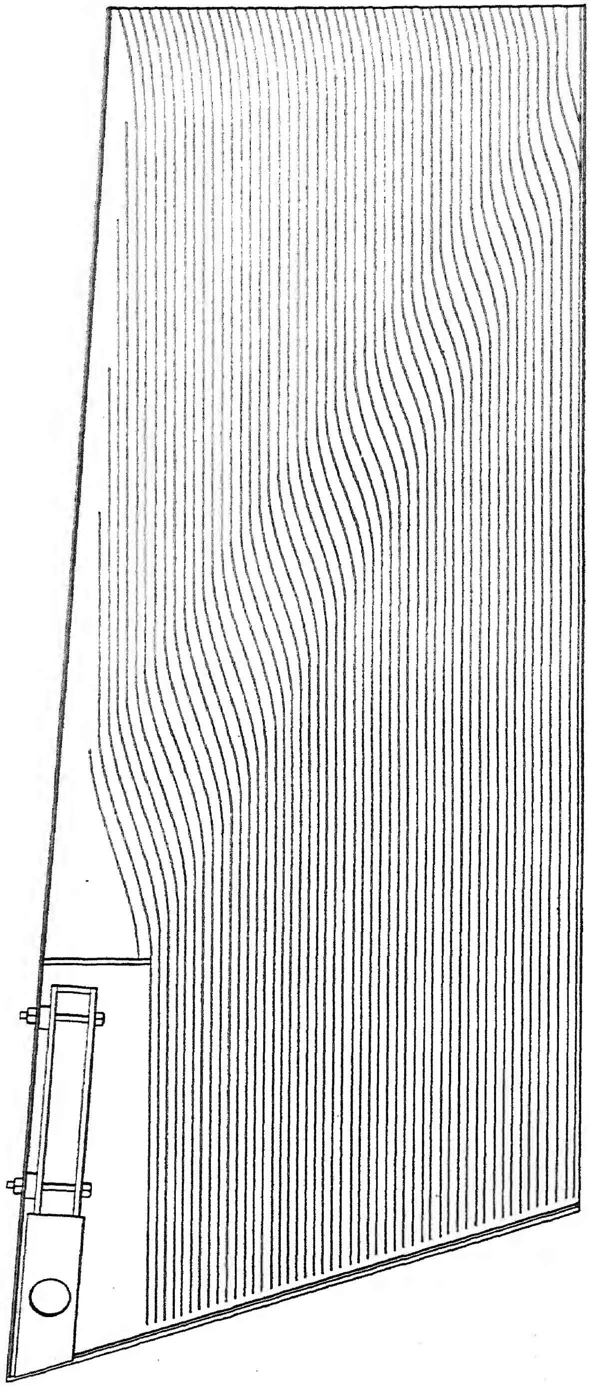


FIG. 2.—THE BUTCHART RIFFLE SYSTEM.

insufficient means for separating the strata. The new system not only stratifies the minerals but is provided with ample means for segregating and separating them. The old system has a single cleaning channel, or zone, consisting of the diagonal depression adjacent to and parallel with the successively advancing ends of the riffles, and this is easily overloaded in a way to defeat in large measure the object desired. The new system has as many individual cleaning channels as there are riffles upon the table, and these are not easily overloaded because they are filled in succession and back each other up.

The old system is not adapted to the handling of very coarse table feeds, stopping for good work at about 2½-mm. size, while the new system handles very large tonnages of what have hitherto been jig sizes, with a much simpler arrangement of machinery and with greatly improved general results.

The new system eliminates the necessity of hydraulic classification beyond the desliming stage. Desliming or classification of any kind is not required on coarse feed because the table rejection is usually dewatered and reground for further treatment, and when either primary or reground material is to be handled it is necessary only to remove thoroughly the slime either in spitzslutte or drag classifiers. When classification is carried further than this too much of the fine sand is often eliminated, with detrimental effect upon the work of the new table.

The new system requires less water than the old. The water consumption of the new system is from 250 to 275 gal. per ton of feed; therefore, when these tables displace jigs the water consumption is reduced from 50 to 75 per cent., and when used instead of the old system it is reduced approximately 50 per cent.

With these advantages the new system is very important in wet concentration, particularly where the ratio of concentration to be practiced is medium or low, and when used in conjunction with a successful flotation treatment of colloid overflows it makes possible the construction of a most desirable wet concentrating plant, adapted to large tonnages in relatively small space, and for a minimum capital expenditure per ton treated.

In developing the table at Morenci, it was first recognized that the Butchart system of riffles gave great stability to the operation of tables under varying conditions of feed and water. Suddenly increasing the quantity of feed would not interfere with the ability of the table to make clean concentrate, and it was not necessary continually to be making adjustments of the table slope in order to bring the concentrate to the proper cutting-out point. These differences were in great contrast to the old system and discounted the indifference plus carelessness of the average operative now found in large mills. This consideration alone was considered sufficient to justify the adoption of the new system

in the Arizona Copper Co.'s No. 6 Concentrator, where this system of riffling has since been evolved into its present form.

At first the riffles were thin and the ultimate capacity of the table was unknown. Discoveries in the matter of capacity came about rapidly through experimentation. For example, it was found desirable to reduce the insoluble material in Hancock jig concentrate, which was a mixed product ranging in size from 3/8 in. down to 200 mesh, containing 15 per cent. copper and 30 per cent. insoluble. The standard No. 5 Wilfley table was first installed for this work, but it was found impossible with this system of riffling to handle successfully the range of sizes or the excessive and variable tonnages coming from the jigs. Butchart strips were then substituted for the Wilfley riffles. These strips were thicker than usual, purposely making deeper channels, and it was found that very little work of adaptation was required to get excellent results. Heavy tonnages could be treated with a minimum amount of material sent to the middlings; the table responded to great variations of load; the insoluble material was reduced to less than 15 per cent.; and the coarse concentrate, instead of working down and arranging itself in the rear of the concentrate band, as it does with the Wilfley riffle, was discharged out of the top riffle into the concentrate launder; and time samples showed that the table load averaged about 80 tons per day.

Experiments were then tried on coarse unclassified feed up to and including the undersize of 4-mesh screen affording square openings of 3/16-in. size. This feed had a concentration ratio of 15 into 1, and it was found that as much as 200 tons per day could be handled successfully, making clean concentrate containing a range of sizes from 3/16 in. to 100 mesh and finer.

The plant was being operated at full capacity during a period of reconstruction and it became necessary to remove the intermediate Hancock jigs and to crush the primary-jig rejections in one operation in 8-ft. Hardinge mills direct to 20-mesh size for treatment on sand tables. A paddle-wheel classifier, which was a fairly good deslimer, was in use following the Hardinge mills. Floor space was limited by construction operations and but few tables could be accommodated at the time, so a few of the new ones were fitted with somewhat thicker Butchart curved strips, and the relatively rich product of the paddle wheel was fed upon them to what appeared to be maximum capacity, as judged by the appearance of the concentrate and panning of the tailing. Time samples then brought out the surprising fact that the tables were often handling more than 100 tons each, which was three times the load of neighboring Butchart machines and five times the load of ordinary Wilfley riffles, and were making concentrate carrying but 17 per cent. insoluble, with averages as follows: Feed, 95.2 tons, 2.10 per cent. copper; concentrate,

17.70 per cent. insoluble, 15.83 per cent. copper; tail, 0.86 per cent. copper; ratio, 12.1 into 1; extraction, 62.4 per cent.

The subsequent adoption of this system in a neighboring mill has greatly simplified the operation in its primary stages and has reduced costs without detriment to final metallurgical results. In making the change 10 of the No. 5 type Wilfley tables were rearranged for use with the Butchart riffles, and eight of these handle the complete ore tonnage with astonishing results, making it possible for the company to so change and simplify its flow sheet as to dispense with the use of a large amount of machinery with its accompanying expense for repairs, power and operation. This is illustrated by the comparative flow sheets (Figs. 3 and 4), which show the equipment used before and after the change, in which the following list of machinery was discarded:

13	42 in. by 9 ft. trommels
4	one-compartment Harz jigs
8	two-compartment Harz jigs
12	three-compartment Harz jigs
1	two-compartment shovel-wheel classifier
14	12 by 12 ft. settling tanks
17	Wilfley tables
4	No. 2 Deister tables
5	No. 3 Diester tables
1	V-dewatering tank
1	elevator
1	hydraulic sizer
2	dewatering boxes
1	vanner
84	

For more than a year the eight primary tables referred to have been handling an average of 1,600 tons of feed per day, the feed consisting of the undersize of 7-mm. screens, going to the tables without desliming, classification, or other preparation, with the typical results—the average of a long period—which follow:

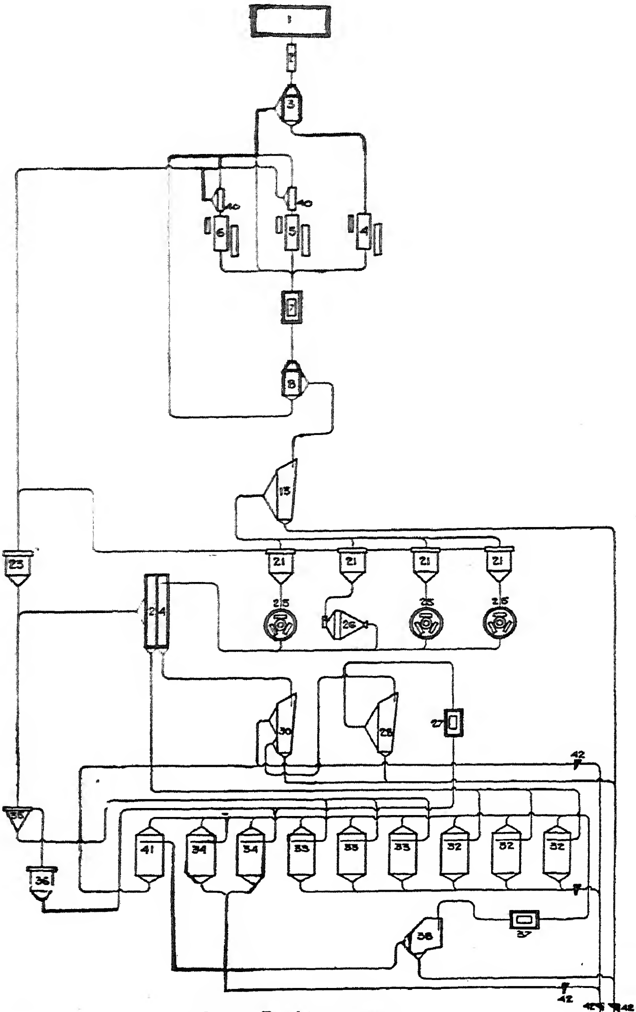
Tables with Butchart Riffles

Head		Concentrate				
Tons Each	Per Cent. Cu	Per Cent. Cu	Per Cent. Insoluble	Reject, Per Cent. Cu	Per Cent. Recovery	Ratio
160	3.15	16.65	9.8	1.42	60.04	8.8:1

Group of Harz Jigs (at same time)

35	3.19	17.43	11.9	1.63	53.95	10.1:1

The reject from the above was reground to 1½-mm. size, deslimed by drag-belt classifier, sand treated on Butchart riffles and same sand

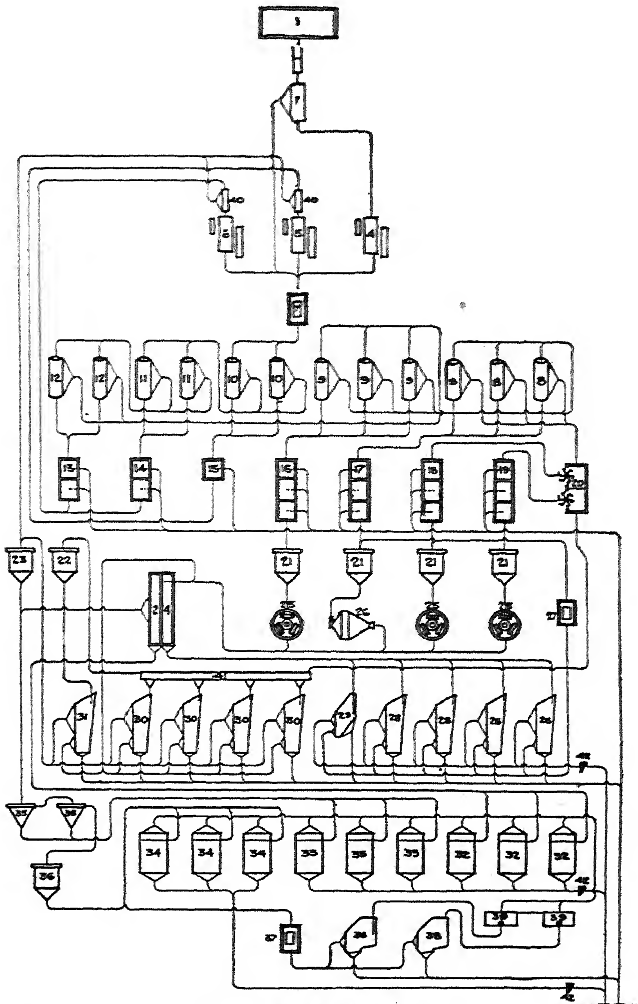


LIST OF MACHINES

Sym	No	Description	Sym	No	Description
1	1	750 Ton Mill Storage Bin	22		
2	1	Plunger Type Ore Feeder	23	4	Wood Settling Tanks
3	2	Zig-Zag Dry Screens	24	1	2-Belt Drag Classifier
4	1	Set 40"×16" Roughing Rolls, To ½"	25	3	5-Ft Chilean Mills
5	1	Set 40"×16" Regrinding Rolls, to 5mm	26	1	36"×8 ft Hardinge Mill
6	1	Set 40×16" Regrinding Rolls, To 5mm	27	2	Middlings Elevators
7	2	Elevators	28	2	Wilfley Tables
8	4	Zig-Zag Screens	29		
9			30	5	National Tables, 4 used
10			31		
11			32	9	Frue Vanners
12			33	10	Frue Vanners
13	5	National Tables - 4 used	34	18	Frue Vanners
14			35	1	V-Dewatering Tank
15			36	6	12×12 Settling Tanks
16			37	1	Concentrates Elevator
17			38	2	#3 Deister Tables
18			39		
19			40	2	Dewatering Shaking Launders
20			41	2	Frue Vanners
21	4	48"×36" Dewatering Tanks	42	4	Automatic Samplers.

Total Number of Machines = 91

FIG. 3.—FLOW SHEET AFTER ADOPTION OF BUTCHART RIFFLE SYSTEM.



List Of Machines

Sym.	No.	Description	Sym.	No.	Description
1	1	750 Ton Mill Storage Bin	22	4	12'×12' Concrete Settling Tanks
2	1	Plunger Type Ore Feeder	23	14	Wood Settling Tanks
3	1	42"-9' Trommel 5/8" ϕ Hole	24	1	2 Comp. Drag Classifier
4	1	Set 40"×16" Roughing Rolls, To 1/2"	25	3	5 Ft Chilean Mills
5	1	Set 40"×16" Regrinding Rolls, To 3/8"	26	1	36"×8' Hardinge Mill
6	1	Set 40"×16" Regrinding Rolls, To 7 mm.	27	2	Middlings Elevator
7	2	Bucket Elevators, 22" Belt 70' Crs	28	11	Wilfley Tables
8	3	42"-9' Trommels, 2½ mm ϕ Hole	29	4	No. 2 Deister Tables
9	3	42"-9' Trommels, 5 mm ϕ Hole	30	4	Wilfley Tables
10	2	42"-9' Trommels, 5/16" ϕ Hole	31	4	Wilfley Tables
11	2	42"-9' Trommels, 1/2" ϕ Hole	32	10	Frue Vanners
12	2	42"-9' Trommels, 7 mm ϕ Hole	33	10	Frue Vanners
13	4	2 Comp. Hartz Jigs	34	20	Frue Vanners
14	4	2 Comp. Hartz Jigs	35	2	V-Dewatering Boxes
15	4	1 Comp. Hartz Jigs	36	6	12'×12' Settling Tanks
16	3	3 Comp. Hartz Jigs	37	2	Tailings Elevators
17	4	3 Comp. Hartz Jigs	38	7	No. 3 Deister Slime Tables
18	3	3 Comp. Hartz Jigs	39	2	Dewatering Boxes
19	2	3 Comp. Hartz Jigs	40	2	Dewatering Shaking Launder
20	1	2 Comp. Shovel Wheel Classifier	41	1	Hydraulic Sizer, 4 Spigots
21	4	48"×96" Dewatering Tanks	42	4	Automatic Samplers

Total Number Of Machines = 159

FIG. 4.—FLOW SHEET BEFORE ADOPTION OF BUTCHART RIFFLE SYSTEM.

simultaneously on standard Wilfley riffling, with the following comparative results of four days' continuous sampling:

One Table with Butchart Riffles

Tons	Head Per Cent. Cu	Concentrate Per Cent. Cu	Concentrate Per Cent. Insol.	Rejections Midd. Per Cent. Cu	Rejections Tail Per Cent. Cu	Rejections Midd. and Tail Per Cent. Cu	Ratio	Per Cent. Recovery
101	1.66	12.46	11.3	0.91	0.55	0.61	11.3:1	66.50

Comparative saving and loss: Middling, 6.94 per cent.; concentrate, 66.5 per cent; tail, 26.56 per cent. Water consumption, 16 gal. per minute, 256 gal. per ton.

One Table with Wilfley Riffling

Tons	Head Per Cent. Cu	Concentrate Per Cent. Cu	Concentrate Per Cent. Insol.	Rejections Midd. Per Cent. Cu	Rejections Tail Per Cent. Cu	Rejections Midd. and Tail Per Cent. Cu	Ratio	Per Cent. Recovery
35.5	1.47	13.21	10.9	2.26	0.59	0.76	17.5:1	51.25

Comparative saving and loss: Middling, 14.75 per cent.; concentrate, 51.25 per cent.: tail, 34.0 per cent. Water consumption, 12 gal. per minute, 867 gal. per ton.

A table with Butchart riffles was afterward tested against a group of five standard Wilfley tables, the latter handling as nearly as could be gauged the same tonnage as the one Butchart, with the following averages for four days' continuous sampling:

One Table with Butchart Riffles

Tons	Head Per Cent. Cu	Concentrate Per Cent Cu	Concentrate Per Cent. Insol.	Rejections Midd. Per Cent. Cu	Rejections Tail Per Cent. Cu	Rejections Midd. and Tail Per Cent. Cu	Ratio	Per Cent. Recovery
96.8	1.57	14.32	17.4	1.00	0.74	0.79	17.3:1	52.58

Comparative saving and loss: Middling, 11.54 per cent.; concentrate, 52.58 per cent.; tail, 35.88 per cent.

Five Tables with Standard Wilfley Riffles

Tons to 5 T	Head Per Cent. Cu	Concentrate Per Cent. Cu	Concentrate Per Cent. Insol.	Rejections Midd. Per Cent. Cu	Rejections Tail Per Cent. Cu	Rejections Midd. and Tail Per Cent. Cu	Ratio	Per Cent. Recovery
115.9	1.39	15.20	16.8	1.55	0.68	0.81	24.8:1	44.07

Comparative saving and loss: Middling 15.99 per cent.; concentrate, 44.07 per cent.; tail, 39.94 per cent.

It will be noted in these tests that the insoluble is slightly higher in the Butchart concentrate. This is because this type of riffling cuts more oxidized copper into the concentrate than the Wilfley system does. The saving of this oxidized mineral at this stage is very advantageous, for it is as high in copper as milling can make it, but being lighter it goes off in the Wilfley system in the middling *and thus raises the copper content of the rejections and increases the ratio of concentration at the expense of the extraction.* It will be noted further that the Butchart riffle system has evidently removed all the free mineral from what would be the middling zone to the concentrate. Of course, the Wilfley middlings could be re-treated and the free mineral removed, but this complicates the operation and is unnecessary when the Butchart system is used. Middling on the Butchart system is a true middling, consisting of grains needing further crushing to free the mineral, together with the lighter part of the oxidized mineral, etc., and with a small amount of free but very fine mineral grains. *The middling on the Wilfley system consists of the above, plus what free and relatively coarse mineral the cleaning zone, on account of its overload, has failed to separate.*

In another plant and district, on a type of ores very different from those of Morenci (having no water-soluble oxidized or carbonate copper), loads as heavy as 200 tons per day of thoroughly deslimed but otherwise unclassified feed, being the undersize from 1½-mm. screens, assaying an average of 1.34 per cent. copper, have been treated with an extraction of 54 per cent. of the copper content, as compared with an extraction of 38 per cent. effected upon tables running in parallel out of the same launder but equipped with a modified Garfield system of riffling. The concentrate in each instance was cut to 20 per cent. insoluble, which is the smelter standard worked to in this case. The rejections of the Butchart table contained very little copper except in the form of attached mineral, while the other tables rejected much free mineral, through inability to clean and segregate it to the concentrate launder.

The modification in the Garfield system of riffling above referred to consisted of cutting off the ends of the riffles diagonally on the concentrate corner of the table to make a cleaning zone there, at the same time retaining the deep channels of the Garfield system.

If the Butchart riffles were to be straightened out, *i.e.*, laid without curves, the result would be the Garfield system of riffling, and it will be realized that the table would then require more side slope to prevent the whole load from going over the concentrate end; and on account of the violent agitation produced when enough side pitch is given to the table to carry the sand into the tailing launder properly, the fine material will not have a good chance to settle and much more of it will go over into

the rejection than would be the case when the curves are used, for with the new system the side slope is such that a gentle rolling over of the load is accomplished. The movement of the load bodily along toward the concentrate end is checked by the curves and these enrich the product passing them to whatever extent the plan of operation requires. Thus the Butchart system makes a superior roughing as well as a superior finishing table. When simple roughing is the work to be done the curves are modified to suit that class of work.

In another district a company has installed a "pilot mill" to determine the best flow sheet adapted to its ores. It has developed in this mill that two Butchart tables handling the drag-belt deslimed product of Hardinge mills, assaying 1.35 per cent. copper, ratio 8.5 into 1, are making concentrate averaging 9 per cent. copper, 8 per cent. insoluble, 32 per cent. iron, and with middling plus tail that will average 0.35 per cent. copper. The extraction in this case is from 75 to 80 per cent. of the copper present.

In this pilot mill the roughing out of the concentrate of the first separation stage above the regrinding mill is done on a Butchart table on unclassified and not deslimed feed that has passed a 7-mm. screen, with the same generally good results as to production of finished concentrate that have been noted in the previous cases.

This pilot mill is proving that with a simple arrangement of crushing machinery reducing the ore to 7 mm. for the first separation by the new system, the rejection through regrinding mills for finished reduction, desliming of the reground sand, treatment of the sand over Butchart tables and of the slime by flotation, a very high percentage of the total value will be recovered. Most of the final rejections are made from Butchart tables and are sand tailing containing the minimum amount of copper noted above, indicating very cheap costs for construction of plant and, on account of its simplicity, very simple and inexpensive operation throughout.

Operation Analysis

In order to understand the marked change in table conditions brought about by this new system of riffling, it is necessary to analyze in some detail the method of operation in the old system and compare it with the new one.

In the work of each system the coarse and more cubical pieces are more free to move and more easily rolled by the carrying water than are the smaller and flatter particles, and their arrangement upon the table is in accordance with the law governing the angle of repose of particles partly suspended in water. The tilt sidewise given to the table causes the wash water to move more speedily and also provides more slope to assist the rolling-over action of the mass being treated, with the result

that the coarser particles of low specific gravity go clear across the bed of pulp upon the table and arrange themselves in the rear of the mass moving forward, while the successively smaller sizes, with those having successively greater specific gravity, tend to arrange themselves farther up on the slope and farther along toward the discharge end. Thus the forces at work tend to arrange the bed upon the table with successively smaller grains from rear to front and successively lighter grains from bottom to top. The differential motion moves the whole mass lengthwise, while gravity is rolling it crosswise the table deck, and the heavy bottom stratum of mineral consisting of medium-sized and fine particles appears first upon the cleaning zone of the Wilfley system, and goes farthest up the slope against the influence of the dressing water tending to wash it back.

Each channel of the Wilfley system properly stratifies its portion of the minerals and then becomes a conveyor to bring the mineral strata under the influence of the wash water in the single diagonal cleaning zone just beyond the ends of the riffles.

When the quantity of mineral of high specific gravity is in a relatively small proportion, the recoverable portion of it is pushed forward in the grooves by the differential movement beyond the terminals of the riffles into this zone, where it encounters the clear water on a new slope and where it is rolled over itself, while the fine sand is eliminated in the process of dressing, the finer and heavier particles finally occupying the thinnest edge nearest the wash-water box, the coarser working down and arranging themselves in the rear of the mineral band. This is the situation desired and the one that results in good separation, but this action cannot take place except where the layer of mineral is sufficiently small in volume so that it can all be cleaned outside and beyond the ends of the riffles; which means that if the separation is to be satisfactory the table must be fed with tonnages affording only the amount of concentrate which the limited cleaning zone can successfully handle, and that if larger tonnages of low-ratio material are handled the cleaning zone will fail proportionately with the overload, the clean-mineral stratum will not emerge from the riffles and will be covered by the middling stratum to a large extent, and therefore will go off as middling, or to the detriment of the concentrate if cut out with that class.

With the Butchart system the entire surface of the table is covered with riffles which are made to perform useful work. Each of the channels becomes a distinct separating device which cleans its own concentrate in the curve between the riffles instead of upon a smooth unriffled surface. It has the ability to perform this function on the upper one-half of the deck, on the large quantities of concentrate produced, the balance of the surface being free to treat the more difficult portion of the concentrate, to clean up middling, and to be ready to handle larger quantities of concentrate when the load or ratio fluctuates.

The riffles extend the full length of the table, are exactly alike as to dimensions, and taper from the rear, or mechanism, end to the front, or discharge, extremity. For average operations they usually taper from a height of ½ in. at the rear to ⅛ in. at the front end. These dimensions are varied to provide such concentrate-carrying capacity as may be demanded by the average ratio of concentration in the material to be treated. They may be, if required, 1 in. in the rear, tapering to ⅜ in. at the tip, or discharge, end. The curved deflection of the riffles together with the transverse inclination of the table produces in the curved channels a downward slope toward the rear, or mechanism, end of the table, and the contents of the channels have to climb up this slope against a gentle stream of water passing down it. As the channels in the cleaning zone are not parallel to the direction of motion, the straight-line movement of the table causes a circular agitation or side shake to be imparted to the contents of this portion of the channel, subjecting all of the contents to a vanning action which causes the fine-silica content of the strata to pass down the channel with the clear wash water and to pass over at the lower end of the curve, and, notwithstanding a great variation in the concentrate sizes, effectually cleans the concentrate while climbing up the slope. At the discharge end of the riffles there is usually another curve, which serves the purpose of preventing too large a portion of the dressing water from falling over the end of the table with the concentrate. This is for assuring a steady and even flow of water completely across all of the parallel channels of the table below the wash-water box, so as to produce a steady and even flow of water down the channels in the cleaning zone. The terminal curves are indispensable when the riffles have a thickness of from 3/16 to ⅜ in. at the discharge extremities of the table.

The stratification and removal of concentrate is so rapid that most of it goes off in the upper one-third of the table deck in 12 to 15 riffles, depending upon the ratio of concentration. The remaining mineral is successively less in volume and is cleaned and brought forward in the larger area farther down the table, and is lighter and successively poorer in grade. Thus there is a large space, between the main concentrate discharge and the silica rejections, which is occupied by a small quantity of relatively poorer concentrate, and there is no large accumulation of concentrate near the middling corner as in the old system. The feed on the middling corner of the table has become so thoroughly impoverished that it is sometimes found necessary to take the contents of several of the lower riffles into the middling.

The effect of the uniform feed and water distribution over this completely riffled table is to produce uniform velocities in the water moving on the two zones each side of the cleaning curves, with an excellent classification of the feed upon the main body of the table. As previously

mentioned, the sands arrange themselves in successively decreasing sizes from the rear toward the middling corner, and the perfection with which this action occurs is found to be an excellent visual guide in judging the work that the table is performing. The better the classifying action noted the better the recovery will be.

In ores affording low ratios of concentration, the sulphide minerals present are usually massive and coarse concentration is usually practiced, and in their anxiety to avoid sliming of the sulphides engineers try to

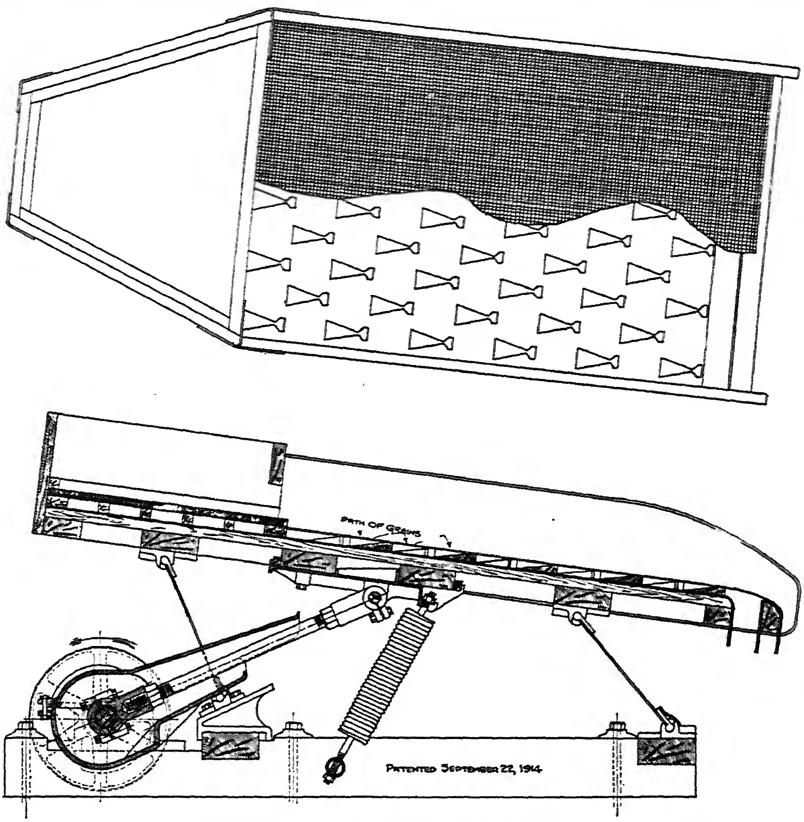


FIG. 5.—VIBRATING SCREEN.

remove them in several stages, each stage adding materially to the complication, cost of construction, maintenance, and cost of operation. The instances quoted in this paper show that it is not always necessary or economical to have several stages in the roughing-out process, and the author believes that it is seldom advisable.

Roll crushing down to 5-mm. size does not require choke feeds and does not slime the minerals excessively. It is the reduction in regrinders that produces slime rapidly. There is no evidence to show that reducing a ½-in. cube of sulphide mineral in crushers or rolls into 10 or more frag-

ments passing a 5-mm. screen, in preparation for a single-stage treatment, will result in the production of more colloidal mineral than is produced in the trommels and jigs of a more complex system using more stages, and in the light of the work accomplished and recorded the proposal to simplify the operation by crushing directly to roughing-table size does not seem illogical.

Many copper concentrators have corrosive water to contend with, wherein all iron parts of the equipment coming in contact with the water are subject to corrosion. This makes maintenance of trommels, especially the finer sizes, very expensive, and is an argument against the complex methods commonly in use.

Water is expensive at Morenci and has to be used with great economy. It is kept in circulation as long as possible. The use of lime is resorted to, but the solutions are still corrosive. Wet screens are necessary, and in order to avoid the high first cost and high maintenance expense for these, a vibrating machine for working wet has been devised and very successfully used. As shown in Fig. 5, it is a small machine, carrying in this case a 5-mesh by 0.054 bronze wire screen. It is made of wood, copper, and rubber, the latter being used wherever water comes in contact with it, and, since there is no scouring action from the ore passing over it, the light wire screen mentioned above lasts approximately 100 days, during which time about 40,000 tons of ore pass through the wires. This is about 10 times the life of ordinary wet screens of this mesh. The machines do not clog, because a sluice deck is maintained close to the under side of the screen cloth, and the coarser part of the undersize strikes against the protruding oversize particles and drives them back. On its way down the slope the water also goes back and forth through the screen in a way which is effective in making the separation rapid and in preventing blinding of the meshes. This machine has proved to be a further factor auxiliary to the simplification of flow sheets through the use of the Butchart riffle system.

The results of the improvements are regarded as important in promising to lessen greatly the outlay required to treat a given tonnage in large plants, and by making it possible to assemble a cheap and effective concentrating plant for small tonnages.

Discussion

R. H. Richards, Boston, Mass.—The Butchart riffle seems to violate, according to our preconceived notions, some of the very essential principles of the Wilfley riffle, and therefore ought not to succeed, but it does. On this account it must conform to other very important principles, or it would not succeed. It would be interesting to know what these principles are which enable it to succeed.

E. P. MATHEWSON, Butte, Mont.—I cannot explain the principles on which this riffle acts, because I have not given it personal attention. All I can say is that I first saw the riffle in operation several years ago at the Steptoe Valley mill in Nevada, and I was told that a man had been around there guaranteeing that these riffles would make the table do twice as much work and do it better than with the ordinary Wilfley riffles. I heard nothing further about it for a long time.

I heard about a year ago, from David Cole, the author of this paper, that the riffles were a success, and that he was using them at Morenci, and if we were interested he would send the inventor to show us how they worked. As we knew of David Cole's reputation, we invited Mr. Butchart to come up and show us how his riffles worked. We did not have sufficient confidence in the invention to guarantee Mr. Butchart his traveling expenses, but he came up entirely at his own expense and put in a set of the riffles on one table. It was known as a roughing table in our No. 1 section. We put it to work, tried it, and had it carefully tested by our testing department, and we found on roughing work it was a grand success, that we could get more tonnage and better work done by this device. We were encouraged by Mr. Butchart to try it on classified feed, and there we found that it did not do such good work, that it was considered no improvement with such feed. We finally made an arrangement with Mr. Butchart to put this device on all the *roughing* tables we had at that time in our mill, which were nine or ten. About the time we had this introduced, we made a new design for our mill, and a new flow sheet. We adopted the Butchart riffle for the entire mill, for our *roughing* tables, and these will be the only tables of the Wilfley type we will have in our mill.

DAVID COLE, Morenci, Ariz. (communication to the Secretary*).—Mr. Mathewson is substantially correct in his remarks, and in way of supplementing his statement I would add that this riffle system was first developed in an experimental way by Mr. Butchart in Los Angeles. After he completed his experiments there he came to Morenci and made the first practical application of the riffle in the No. 6 concentrator of the Arizona Copper Co. This was in the latter part of 1911. He carried on a long series of experiments at that time and these led to the adoption of the riffles by Mr. Carmichael for the 22 tables then in use. Experiments with the new riffle system were also carried on at the same time in the Detroit Copper Co.'s mill here. One or more tables were riffled for experimental purposes and operations were carried on under the supervision of the company's metallurgist, but the Detroit company did not adopt the riffle until two years later, after it had been perfected in the Arizona Copper mill, as mentioned in my paper.

Subsequent to these original applications of the riffles at Morenci, Mr.

* Received Mar. 29, 1915.

Butchart carried on some experiments in the Steptoe plant of the Nevada Consolidated, and I understand that one section of the mill was riffled with his system. This worked so well that the company purchased the right to use the riffle in its entire milling plant.

All of the tables equipped with the Butchart system of riffles up to the time above referred to carried thin strips, which were applied to tables handling relatively fine feed; that is, the field of the new system of riffling was not extended beyond that of the field of the regular Wilfley riffle. No attempt had been made to handle feeds coarser than had been found to be adapted to the Wilfley system. It was found at that time that the new system would handle larger tonnages than the Wilfley and, as mentioned in my paper, the tables were much more easily looked after, but the ability to handle very large tonnages of very coarse, unclassified feeds had not at that time been developed.

When I assumed the responsibility of remodeling the Arizona Copper Co.'s No. 6 concentrator about 2½ years ago, I found the riffle in use in the mill and was favorably impressed with its work. I thought I saw possibilities of improvements and I advised the company to purchase the right to use the riffle in all of its table work instead of only the 22, and this was done. Mr. Butchart then came to Morenci and remained several months working out the improvements to the successful issue mentioned in my paper.

In July, 1914, I visited Montana with reference to advising the Anaconda company as to changes contemplated in their Washoe concentrator, and while at Anaconda I put a set of Butchart riffles on a Wilfley table doing roughing work in the No. 1 section, which had been remodeled so as to use the "Great Falls" system of concentration, described by Mr. Wiggin in his paper read at the Butte meeting. I operated this table for several days with excellent results. After returning home I communicated this fact to Mr. Butchart, who expressed a desire to go to Anaconda at his own expense to see the table in operation there, and, if the management should desire him to do so, to carry on further experiments. This led to the arrangement and results mentioned by Mr. Mathewson.

The new system of riffling is not well adapted to handle classified feeds. Hydraulic classification, carried out in the conventional manner common in concentrating practice, does not separate the feeds properly for the Butchart riffle. The coarser the feed the more necessary it is to have mixed sizes. For primary work not making tailings no classification or desliming is necessary, and for conventional table sizes when making finished tails it is, as I have pointed out in the paper, only necessary to remove the very finest sand together with the colloidal slime, for the table does the classifying after this and it requires the large range of sizes in the feed in order to hold the mineral and prevent losses; for all of the fine mineral would not have been taken out by hydraulic classification—

some of it would go with the table feed and would not be held on the table except in the presence of suitable size sand.

Hydraulic classification is in fact rough concentration and hydraulic classifiers are rough concentrators, the use of which is very largely displaced by the use of the Butchart system of riffles.

Concentration engineers and mill men are slow to accept this feature of the situation because they have been so thoroughly drilled in the neces-

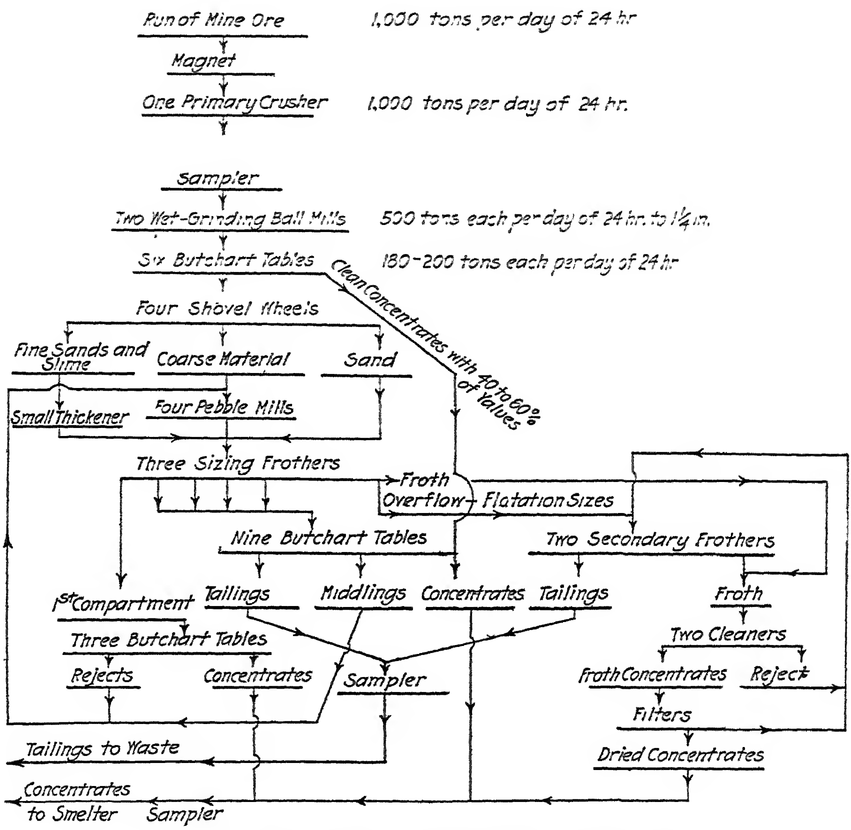


FIG. 6.—FLOW SHEET FOR LOW-RATIO COPPER ORES.

sity for screen sizing and hydraulic classification that it has become a sort of religion in the trade and to change it is almost heresy. They regard the old order of things as law, but the operation of the Butchart riffle proves these ideas to be largely erroneous.

Very great improvements have been made in the past year in crushing, sizing, elevating, and concentrating machines. These improvements have had for their primary object the reduction of all divisions of costs through the reduction of stages, thus simplifying and cheapening the concentration process. The Butchart riffle has aided materially in

carrying out this program and with the marked improvements in crushing and screening machinery above mentioned we are now able to dispense with a large number of the operations previously thought absolutely essential. We can now crush the ore from run of mine to the first Butchart table size by one pass through a primary crusher plus one pass through a ball mill running wet. We can put the pulp so prepared direct from the ball mill on Butchart tables *without any classification or de-*

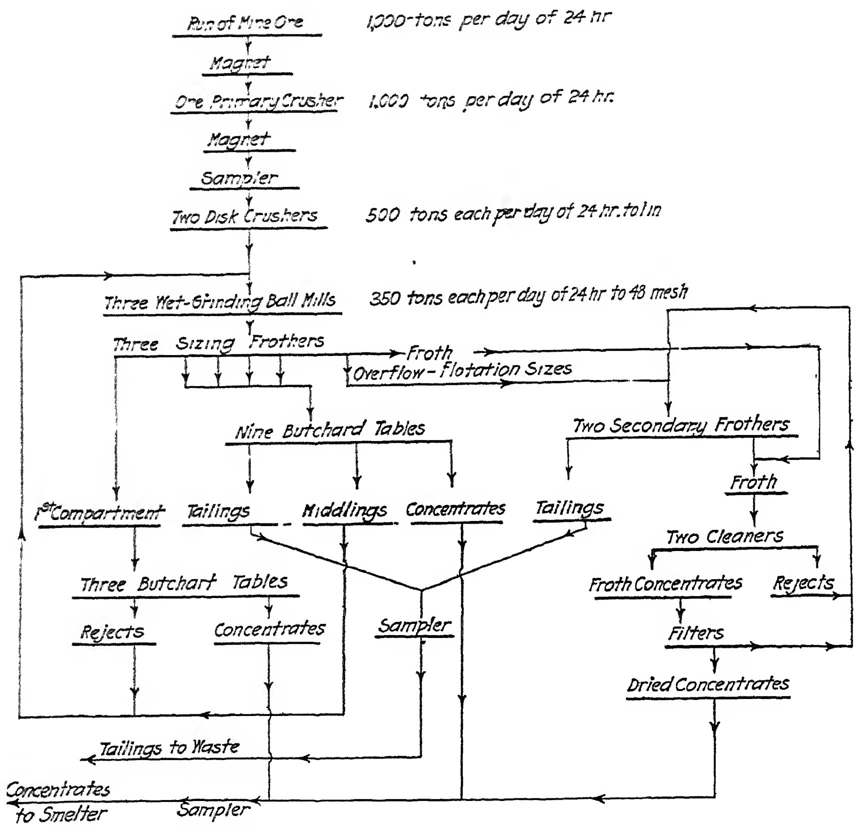


FIG. 7.—FLOW SHEET FOR HIGH-RATIO COPPER ORES.

sliming. We can separate on these tables, in a condition to go directly to the smelter, a very large percentage of the minerals present. This concentrate will contain sizes from ¼ in. to 200 mesh, in a clean condition. The colloidal copper will pass over the side of the machine with the slime rejections and if desirable may be separated by suitable mechanical classifiers so as to by-pass the final crushing machine. Demonstrations now being carried on here indicate strongly that the final regrinding mill will be most efficient and desirable when run in a closed circuit with a frothing sizer, the latter having capacity to complete three important

operations: first, pass the colloidal copper sulphide, the so-called slime or floured flake copper, over the top of the machine with the froth; second, remove the oversize which has escaped from the regrinder, in a condition to be returned to same; third, prepare the sands already sufficiently fine so that the feed will be ideal for table concentration—this will be the *natural* result when the slimed copper is removed by the frothing and the flotation sizes are removed with the overflow of the machine. This overflow will be treated by further frothing if necessary. With this arrangement the operator of the frother has control of the size at which the pulp is allowed to leave the closed circuit, and the advantage in allowing the mill to make a quantity of oversize will be obvious, for when held down to grinding all of their product to pass the limiting screen without oversize the capacity of these grinders is greatly reduced.

In the above arrangement there is no colloidal or floured flake sulphide in the feed prepared for the tables—this all goes over the top with the froth or with the flotation sizes—and thus the slime "bugaboo" is removed from the mill man's list of troubles.

These improvements are important and far reaching and mean that we can easily double the capacity, under the same roof, of most copper concentrators now in operation and can construct new plants in much less space and for a very much less capital expenditure than was formerly required. I do not hesitate to say that however much we dislike to do it, we will undoubtedly abandon the old and complex system, with its multiplicity of screens, hydraulic classifiers, settling tanks and slimers, for with the addition of flotation treatment of the slime together with the great improvement in milling mechanism above referred to, very greatly simplified flow sheets (see Figs. 6 and 7) and very simple methods of treatment for the concentration of ore, particularly of copper ores, are demonstrated as now possible, and, in the opinion of the writer, these methods will quickly be recognized and adopted.

German and Other Sources of Potash Supply*

BY CHARLES H. MACDOWELL, CHICAGO, ILL

(New York Meeting, February, 1915)

Up to 1909 the American public had little knowledge of, or interest in, potash. Some remembered that it had to do with soft soap and sore throat, but further they knew not. In 1909–10, the German-American potash war, and the publicity attending hostilities, brought home to America the knowledge that potash was an important article of commerce; that Germany monopolized its production and distribution, and exercised all the prerogatives attending complete control. The present war, with its transportation embargo, has again directed attention to Germany's strength as a potash producer and her present unfortunate situation as a distributor. As a consequence, the search for potash takes on new life and it again becomes a subject of general interest.

My first knowledge of potash came from a study of the German potash exhibit at the World's Fair in 1893. This investigation led to the recommendation that Armour & Co. engage in the manufacture of commercial fertilizers and the fertilizer branch was organized early in 1894. From then to now "potash" has been a live subject.

Two and eight-tenths per cent. of the earth's crust consists of potassium oxide. It is an important constituent of feldspar, granite, and other igneous rocks. Potash is valuable commercially in a water-soluble form. In igneous rocks it is locked in so tight that it is of little value and no utilization in a commercial way has been made of these insoluble forms. Potash in water-soluble form is found in large deposits in Germany; in smaller deposits in Austria and Spain; and in brines in some of our Western arid basins. The deposits in Germany, Austria, and Spain are concentrations from sea waters containing potash derived from the decomposition of potash-bearing rocks.

It has been found that the amount of potash remaining in decomposed rocks is not the same as that in the original rock. For instance, granite containing originally 5.58 per cent. potash contained only 1.49 per cent. in the decomposed rock. Syenite containing 6 per cent. in the original

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rock contained only 0.23 per cent. in the decomposed rock. Soil also loses potash.

The probable explanation for this loss of potash is the passing of the rain and soil solutions over constantly renewed exposures of the rock, washing out the soluble portion of the potash, which eventually finds its way into the rivers and finally to the sea. It is estimated that one cubic mile of the river waters of the United States contains 20,358 tons of potassium sulphate. It can thus easily be seen that an enormous amount of the available soil potash is being transported by the rivers and deposited in the sea. The sea brines containing salt, potash, magnesium, and other salts, on concentration drop out the less soluble salt, the mother liquor containing the potash accumulating and later desiccating or crystallizing as veins through rock-salt deposits, over which are formed water-proof covers protecting the deposit and holding it for future use. This briefly is the legend.

German Deposits and Mining Methods

Potash salts are found in Germany in the Stassfurt, the Hannover, the South Harz Mountain and the West Alsatian sections. Mines are also operating close to Hamburg and Bremen, and lean deposits, with poor cover, are said to extend into Holland. The deposits near Muelhausen are within 25 miles of the French frontier. In the Stassfurt and Hannover sections, the deposits are often found in salt saddles of great depth. The cover has been disturbed and water dangers are present. Several shafts have been lost by flooding. In the South Harz and Alsatian sections the layers are more nearly horizontal; the cover has been little disturbed and water troubles are less likely. The depth of the potash and salt deposits from the upper to lower stratum is some 5,000 ft.

Prospecting is done by core drilling. A saturated salt solution is used for drilling, which permits the extraction of a full-sized core of the soluble salt. This method was first used in locating the potash beds of the Aschersleben mine, one of the oldest and largest producing mines. Potash beds have been uncovered in prospecting for oil, coal, and gas. Boring companies are formed, acquire mining rights, prospect, and dispose of proved properties to companies who sink shafts and install concentrating plants.

In Prussia the land owner originally had all the mining rights, while in Hannover the mining rights belonged to the first discoverer of the salt. These laws, of course, have been greatly modified in developing the mines in recent years. A royalty of 60 pfennig per 1,000 kg., or say 15c. a metric ton crude, is generally paid the owner of the rights. Operating companies are organized either as Gewerkschafts, with 1,000 *Kuxen* or shares of no par value, assessments being made from time to time to complete the work under way; or as Actien Gesellschafts, with a fixed capital in shares of a par value of 1,000 marks.

In shaft sinking the Kind-Chaudron system is generally used. Freezing is resorted to if quicksand is encountered. Shafts are sunk by contract. The average cost of shaft sinking is about $150 per foot. Shafts vary from 300 to 900 m. in depth.

The first potash discoveries were made in Stassfurt. The Stassfurt salt mines are first mentioned in literature in 1227. In earlier years they were used as sources of common salt. Early in the 19th century they were sold to the Prussian government, which started the first bore hole in 1839, striking salt in 1843. The first water that was pumped up from this bore evaporated to about 99 per cent. salt, but it soon became very impure with magnesium and potash salts. The first shaft was commenced in 1852 and struck the salt layer in 1857. It was not until 1860 that Professor Francke first advanced the idea of the various strata in salt mines being of importance for other products than salt, and it was in this year that potash was first prepared.

My first visit to a potash mine was in 1906—the mine Sollstedt, near Nordhausen, shipping crude salts and manufacturing muriate. No sulphate was made there because the salts contain no kieserite. Some notes taken at that time may be of interest:

"Shaft sinking began October, 1902; finished June, 1904; depth, 705 m.; diameter, 8 m.; iron tubing in sections to salt layers, wood below (many mines use brick through salt layers). Mine dry. Hoisting engine Magdeburg make, 1,000 h.p.; cages double decked; shaft head, iron with concrete thrust blocks. Takes five minutes to descend. Considerable jump to cage at lower levels due to cable stretch. Ventilating fan run by horizontal compound engine of 10,000 cu. ft. per minute capacity. Engine made at Aschersleben, novel valve motion. One direct-connected D. C. motor for lighting and electrical power purposes. Power requirements, 1,600 h.p. all told. Burn lignite; mined in neighborhood; little smoke. Separate ventilating tunnel and shaft connecting with main shaft about 75 ft. from bottom. Cover rock largely reddish sandstone. There is a layer of salt clay, anhydrite, and rock salt above the potash deposits, protecting the latter from water danger. Color of potash salt (hardsalt), reddish interspersed with blackish anhydrite and yellow rock salt, these latter layers an inch or so wide. Average potash contents of veins 16.5 per cent. K_2O; thickness, 10 m. Salt hard and dry. No timbering necessary. Mining under government control. Large chambers laid off with 5-m. supporting pillars. Chambers are cut out to height of 8 ft., width of 20 ft.; potash blasted down and transported in 1-ton cars to shaft. Pillars are permanent, chambers filled in later with mill refuse. Electric and hand drills used. Cars are hauled up grades by electric traction. Mining rights purchased from owners of land on royalty and percentage-of-profit basis. Mine controls right 2.5 miles in three directions. Property abuts Bleichroder mine owned by Prussian government. Deposits figured sufficient to last 800 years.

"Shaft ½ mile from mill, located on railroad; transportation, drag chain on industrial track, chain dropping over bail of car; no clutch necessary; speed, 2½ miles an hour. Salt, when elevated, conveyed to mill building; dumped to screen over crusher, car being turned upside down. Movable screen bars about an inch apart. These bars work for an eccentric shaft; screen out fine particles and move coarse pieces to crusher. The crushed salt goes to the cookers for concentration, or to the mill for shipment as crude in bulk. Cookers are steel tanks on order of pressure tanks with

agitators. The potash is dissolved in a hot solution of magnesium chloride, known as the mother liquor, leaving the salt undissolved. The dissolved hot solution is concentrated and run to large crystallizing tanks, where, on cooling, the potash crystallizes out, the mother liquor being pumped out, heated by exhaust steam and used over again. The waste salt is used for mine filling. Iron crystallizing vats are arranged in clusters of four, surrounded by cement-covered aisles in which are laid industrial-railway tracks. These crystallizing vats cover a considerable acreage. The crystals are shoveled from vats into cars, then to a washing and draining platform, then to storage. Concentrated salts are shipped in bags; mine-run salts in bulk."

This description answers well for other mines, except that in carnallite mines the mother liquors are not used over again and must be discharged into sewers to go to the rivers. On this account, carnallite mines can only manufacture a certain predetermined tonnage of concentrated salts; their final liquors are metered, and operations must be curtailed when the river water attains a certain hardness. Carnallite mines make the bulk of the sulphate of potash manufactured, owing to the presence of kieserite necessary for the production of sulphate. Some mines have chemical manufacturing subsidiaries, where electrolytic and chemical processes are used to manufacture other forms of potash and potash combinations.

The method of mining varies according to the way the strata lie in different mines: Where the layers are anywhere near horizontal, continuous ordinary stoping is resorted to, and they usually drift on the bottom of the seam, blasting down from the top and proceeding in all directions from the shaft. Some mines, however, are operated by drifting at the top surface of the stratum and digging out underneath the level. Where the various layers approach the perpendicular, they sink shafts in the material, using inclined skipways to the main shaft. When the deposits are in lenses, "gobbing" is resorted to, filling in with common salt. Power houses and other plant buildings are solidly and expensively constructed. The workman's comfort and safety are thoroughly looked after. Government supervision is everywhere apparent.

Cost of Mine and Concentrating

Plant.—A mine encountering no unusual difficulties in shaft sinking, and with a shaft depth of say 750 m., with plant site, rail connection, power plant, chemical plant for concentrating, workmen's houses, and all necessary equipment, would cost from $1,250,000 to $1,500,000. A second shaft would cost about $350,000 additional. A fully equipped two-shaft mine, of an annual capacity of say 12,000 to 15,000 tons pure potash, would cost $1,750,000 to $2,000,000.

Cost of Production.—This naturally varies with the quantity and kind of salt mined. Owing to the large number of producing mines, I would say that 36 hr. a week would be average working hours, so the output per

mine is small. The cost of mining carnallite averaging 9 per cent. K_2O and delivering it to the mill is approximately 4 marks, or say \$1, a metric ton of 2,204 lb.; hardsalts (16 per cent. K_2O) would cost 5 marks, or \$1.25 per ton. I estimate that a good mine, producing 7,500 tons of pure potash per year, can produce muriate of potash (50 per cent. K_2O) at \$19 per metric ton; kainite (12.4 per cent. K_2O) at \$2.40 per metric ton. On a production of 20,000 tons pure potash a year, the same mine could produce muriate at \$14 and kainite at \$1.60 per metric ton. The overhead charges are included in both estimates. All costs are f.o.b. cars. These costs are probably lower than the average mine cost under present conditions. Some hardsalts mines, on 24 hr. a day mining, claim to have made muriate as low as \$10 per metric ton. Restricted mining and short working hours now prevent any such costs being attained. Freights to coast during river navigation average about \$1.50 per metric ton, and to American ports \$2.50 to \$3; or say \$4 to \$4.50 per metric ton from mine to American Atlantic ports.

Commercial Conditions

Aside from remote water danger, potash mining is unusually safe mining.

Some years ago, because of overproduction, and with the hope of discouraging new mines on account of expense, a two-shaft law, intended for coal mining and in the interest of safety to miners, was made to apply to potash mines, a reasonable time being given old mines to sink second shafts. The result was unsatisfactory, as old companies would split their holdings, form new companies, sink shafts, and demand new quotas.

For many years, overproduction was the rule and profits the exception. Prussia and other States owned mines. Finally a selling syndicate was formed, all of the then producing mines, including those government owned, becoming parties. Participations in the total sales were agreed on, percentages being figured in thousandths per cent. The syndicate had a life of five years and was headed by a government official. Monetary penalties for violations were exacted. A new syndicate had to be arranged for six months before the expiration of the old syndicate. New mines opening during the syndicate term were taken in and new percentages fixed. Occasionally new mines outside the syndicate would sell below syndicate prices, generally to Americans, but would later enter the syndicate.

On June 30, 1904, one mine refused to enter the new syndicate and for several hours sold freely of bulk salts for five years, finally entering the syndicate subject to keeping such sales as had been made. These sales disturbed the syndicate and steps were taken to make such selling and buying unpopular and unprofitable. Prices on bulk salts were lowered. These reductions automatically lowered the price of the outside mine

contracts, as prices were guaranteed. Five-year exclusive contracts were put in effect. Those having bought outside bulk salts were not to be given lowest discounts on bagged goods. They were to be punished financially for having made outside purchases. Secret discounts were given certain favored buyers. Some of the offending Americans "stood pat," and refused to be punished. In due time a new mine opened up and sold these Americans at lower-than-syndicate prices and for a long term of years, thus defeating the syndicate plans.

In May and June of 1909 I visited Germany and made a thorough inspection of several mines, with the idea of purchase. At that time the various syndicate members were negotiating for the renewal of their syndicate, which expired June 30 of that year. Rumors of serious dissentions were prevalent and the prophecy was made that the syndicate would not be renewed. There were too many mines and too little business. The prophecy proved true, as, at midnight of June 30, members could not agree and free trading was in order. There were several American buyers in Berlin, and, for two hours, a considerable business was effected by seven mines at a material reduction from old syndicate prices and for a term of years. The Prussian government then exerted sufficient influence to stop further trading, for the time. It immediately became known that a large business had been done and mines not participating began to exert pressure to force sellers to cancel their contracts. Thus began the potash war of 1909–10. A Cabinet member discussed the situation in the Reichstag. The press took up the refrain and the stage was set for an interesting commercial comedy. The first suggestion was to levy an export tax on these particular American contracts, but that was decided to be too openly discriminating. One of the contracts contained a clause reading, "Any import or export tax or other governmental charge should be paid by the buyer," meaning any uniform tax, applicable to all business. Here was a legal peg: "Other governmental charge." A law was introduced, and finally passed, declaring potash to be a monopoly; that every producer was entitled to a certain percentage of the total business (the percentage to be fixed by governmental agencies), and that any producer selling more than his fixed percentage should pay a super-contingent tax amounting to more than the purchase price of the contracts. This, of course, only applied to mines selling to America, who had sold large tonnages, and could only apply to American purchasers. Our State Department protested vigorously, and a tariff war was threatened. The situation was strained. For some months these taxes were paid by the buyer under protest. Finally the Americans accepted the situation and a peace was concluded. The amount involved ran into many millions. As a part consideration for canceling the contracts, a percentage of the tax paid under protest was refunded American buyers by the German government. Friendly relations now exist.

During the hostilities, I made the suggestion to the Washington authorities that our government should make a search for potash. Appropriations were made, and the investigation is still under way.

On July 1, 1909, there were 52 producing mines in Germany. Our latest reports are that 160 mines are now producing and from 30 to 40 are under construction. This condition has naturally followed the monopoly legislation and fully $150,000,000 has unnecessarily been invested in the German potash industry. Thirty of the best mines could take care of the world's needs.

United States' Consumption of Potash

Statistics of the year 1912 show 11,164,000 tons of crude potash mined in Germany. This is equivalent to about 1,100,000 tons of pure potash. Of this, the United States takes from 245,000 to 250,000 tons. The fertilizer industry uses from 235,000 to 240,000 tons and the chemical industry, 12,000 to 15,000 tons. The average monthly arrivals have been about 20,000 tons. From Aug. 1 to Dec. 1, 1914, some 10,000 tons of pure potash have come in, as compared to a normal receipt of 80,000 tons. Burlap bags are now becoming scarce, and concentrated salts will have to be shipped in casks, or taken in bulk, if shipped. The outlook is not favorable for heavy winter shipments. The fertilizer manufacturer probably has enough on hand to carry him through his spring season on a restricted-percentage basis. The chemical manufacturer is in about the same situation. Should the war continue through the summer, the present rate of shipment, if continued, would take care of only a small part of our necessities.

Other Sources

Austria.—Small, partly developed deposits exist in Galicia, Austria, near Kalusz. Not to exceed 1,000 tons a year of pure potash has been produced—Austria importing the bulk of her requirements from Germany. It is probable that German influence has prevented the development of these deposits. It would not be good sense for any great investment to be made in Austria to produce potash when one considers the close relations existing between Austria and Germany and the large investment Germany has made in the potash industry.

Spain.—Some three years ago, we received from Europe a mining engineer's report on a potash discovery made in connection with some salt developments in Spain, near Sauria. Later the discovery became public. Analysis of the salts found showed a good proportion of potash, fairly amenable to refining. Mining reports and maps submitted to us led me to believe that, owing to shallow depths and poor cover, shaft mining in the deposit reported on would be dangerous. Brining might, however, be resorted to. Since that time, further discoveries have

been made; concessions have been granted to companies to prospect; and Spain may become a potash-producing nation. That country should be a good distributing center, and, if the government does not, by export taxes or other restrictive measures, kill the goose that lays the golden egg, Spanish potash may come on the market. This cannot be for several years, however, and does not help in the present predicament. The potash seems to be there.

Peru.—I have examined reports showing fair-sized and workable deposits of nitrate of potash in Peru. One of these deposits has been worked in a small way. It is close to transportation and should be further investigated.

American Possibilities

Some potash can be made as a by-product from other manufacturing industries. Practically no progress has been made in this direction in this country.

The sugar-beet tonnage in the United States totals 5,147,000 tons. Sugar-beet wastes from this tonnage could produce 30,900 tons of potassium carbonate, equal to 15,440 tons of pure potash.

Raw wool contains up to 20 per cent. potash salts. The wool clip of the United States weighs 298,000,000 lb. In wool scouring 7,450 tons of potassium carbonate or 4,470 tons of pure potash is washed out and lost. This could be largely recovered. Much raw wool from Australia and other countries is imported, and the potash from the handling of this wool could also be saved.

Potash from Cement Burning.—Many cements before clinkering contain small percentages of potash, say 0.1 to 0.25 per cent. In burning this is volatilized. While the percentage is small, the tonnage is so large it figures quite a production on paper.

Many experiments have been made to recover this volatilized potash. It has been worked out to a point where the physical part of the operation is a success. It consists of condensing the volatilized potash in chambers and dust collectors similar to the bag house used in smelters. Up to the present time, however, no cement manufacturer has produced potash as a by-product. Possibly the investment expense attached to its recovery figures against it. I have no data on this.

The industry is a possible source of a considerable tonnage of potash and in case of necessity can be utilized without any extensive changes in present methods of manufacture. Possibly cheaper recovery methods may be found.

At one time considerable carbonate of potash was derived from the ash from burning cotton hulls. These are now used for fodder, so no potash could be obtained from that source.

A few years ago tobacco stems were distilled destructively for nico-

tine and other products. The ash from this material contained high percentages of carbonate of potash. This method of handling has now been discontinued, but the potash in the ground tobacco stems is saved and used for fertilizing purposes.

This about exhausts the by-product possibilities as far as I know.

Hard-wood ashes and other ashes are gathered and used in fertilizing for their potash contents. City garbage contains appreciable percentages of potash. This garbage in many cities is recovered and used for fertilizing purposes.

Government Work

Following the 1910 appropriations, the Geological Survey and the Department of Agriculture immediately began their work. One of the first suggestions made was that the giant kelps of the Pacific Ocean might be a source of supply. As far back as 1904 the manager of our Pacific Coast fertilizer business called our attention to these kelps and their large potash contents when figured on a dry basis. He proposed to burn them, recover the nitrogen and iodine, and use the ash. He went as far as to gather a few tons of kelp and burn it. Later we did some research work in Chicago, not only on the Pacific but also on Atlantic Coast kelps, with a view to complete utilization, with rather satisfactory results. We decided, however, at the time that the cost of gathering the wet kelp and handling it made it a rather difficult thing to figure out profitably; that we were dry-land people and a fleet of boats would be needed to gather any tonnage; and that there was little chance of protecting any process we might develop. Further, it was so easy to get the German product, that the matter was dropped. The total tonnage is no doubt large, but the kelp has a small per-ton value when wet and its gathering and preparing present a problem difficult to solve with financial success. A McCormick may come along with a kelp reaper, presser, and binder that may give the solution. The government has done splendid work in charting the beds and in obtaining other data. These beds are more or less continuous from Vancouver to San Diego, with several important concentrated areas, one of which is off the port of Los Angeles and another off the coast of Seattle.

Authorities differ as to the relative importance of these two beds, but apparently those along the northern coast are more important than those to the south.

Little real work has been done in the harvesting and utilization of this kelp. Several stock-selling companies have been formed, and a company operating at Long Beach, Cal., is producing about 500 lb. of combined salts per day. The general plan has been to cut the kelp and allow it to wash to shore, and although some success was obtained with a harvesting boat, no attempt has been made toward extensive operations.

The question has been gone into quite thoroughly from time to time, and unless definite outlets can be found for by-products such as iodine, tar, and so-called gums, it is doubtful if the utilization of kelp can be made commercial.

A second source of potash on which the government has reported is Searles Lake, Cal. At this point there is a large dry lake of several square miles in extent, upon which a company is now erecting an experimental plant for the production of the various salts found in the brine underlying the surface.

This brine is pumped from wells sunk through the solid crystals and contains about 35 per cent. solids, 5.35 per cent. in the form of potassium chloride. The plant which is now under construction is expected to be in operation by Jan. 1, 1915, and this plant will produce daily: 5 tons of potassium chloride; 5 tons of carbonate of soda; 2½ tons of borax; 15 tons of common salt; 10 tons of sulphate of soda.

The company has in mind an ultimate plant that will be 100 times this capacity, to be constructed in units similar to the plant now under construction, so that it will be some time before any considerable quantity of potash will be produced. The supply of brine is apparently large, and under the proposed plan should last for a good many years. The entire success of the undertaking would seem to lie in the outlet for the various salts produced, besides potash. I have followed the development of this project from its inception and it looks to me as if it should work out and become an important source of supply. The development is in strong hands financially.

Alunite.—A deposit of alunite located near Marysvale, Utah, is described in *U. S. Geological Survey Bulletin No.* 511. Alunite is a double sulphate of potash and alumina with the following theoretical composition: Al_2O_3, 37 per cent.; SO_3, 38.6; K_2O, 11.4; and H_2O, 13 per cent. The Custer-Marysvale deposits are quite pure for large deposits and rather closely approximate the theoretical. A calcining process followed by leaching releases about 85 per cent. of the potash as sulphate, as reported by the government. Large quantities of hot water are required for leaching. Owing to transportation difficulties and to the necessity of finding a market for a big tonnage of impure alumina, the property has not yet been developed. Large capital expenditure would be required, which would hardly be justified from the potash standpoint alone.

Potash from Feldspar.—The first recorded work on the recovery of potash from feldspar was done by Tilghman about 1845; the first United States patent was obtained by Bickel in 1856. Various investigators have been working in this field practically ever since, receiving stimulus from time to time as the potash question became acute.

In recent years, research on this subject has been along four general lines: First, by the use of natural agencies, such as heat, bacteria, carbon

dioxide, etc.; second, wet chemical methods, such as solution in alkalies or acids; third, dry reactions in which the potash is volatilized; fourth, dry heat reactions in which the potash is converted into a water-soluble form. All of these methods have been more or less successfully operated in the laboratory, but up to date their cost has been prohibitive to commercial development.

The fact remains that there is practically an unlimited source of potash in this country in the igneous rocks, particularly those of the feldspathic type, and no doubt eventually some of these processes will be made commercial, probably through a proper usage of the by-products produced in the recovery of the potash.

The use of feldspar in connection with the manufacture of white cements seems to me to be the most encouraging. An Austrian process looks rather good as far as it has gone. The cost of mining feldspar is considerable. A cheaply mined deposit and the necessity of having this deposit in a locality suitable for the manufacture and distribution of cement would be the controlling features in such an industry.

Leucite.—The leucite hills of Wyoming contain large quantities of potash in an insoluble form. As far as I know, no serious work has been done on this material. It is claimed that by a simple process the potash can be made available for plant food, but in the form offered it is not soluble in water and could not be used in the fertilizer industry, as the various State laws require water-soluble potash. Dana gives an analysis of leucite from the Albani Mountains, containing 21.48 per cent. of K_2O. The leucite hill deposits, according to an analysis from the Armour laboratory, contain 11.34 per cent. K_2O. Other analyses of different samples show even higher potash contents. The quantity is practically inexhaustible.

Other Brines.—The brines of several alkali lakes and ponds in western Nebraska contain appreciable percentages of K_2O. A government analysis of one of these showed 3 per cent. K_2O in the water. An analysis of a sample of the solids from one of these lakes made by the Armour laboratory showed 15 per cent. K_2CO_3 and 33 per cent. K_2SO_4. The Geological Survey reports in substance:

> A number of alkali ponds and lakes existing in Cherry, Sheridan, Morrill, Garden and Boxbutte Counties, Nebraska, have been examined and in some of these the percentage of potash is as high as 30 per cent. of the soluble material. Although the total potash in solution is large it is disseminated in the muds so that profitable extraction will prove difficult. Some of the deposits are near transportation.

Within the last two years, well drillers in Texas have reported the presence of appreciable quantities of potassium in brines at depths around 800 ft. Dr. Carl Scholz, of the Rock Island Railroad, reports that from these drillings he is led to the belief that east of the western boundary

line of Oklahoma, in the vicinity of Magum, a potash deposit may lie within easy reach.

Development work, principally deep-well drilling, is also being done in various sections of the country, especially in the West and Southwest. In most cases the object is to find dried-up lake beds containing soluble potash salts or brines from these sources as well as those draining potash-bearing rock territory. The commercial utilization of such workings will depend, of course, upon the extent of the potash-bearing areas and their concentration.

Summary

In by-product lines, the sugar manufacturer and the wool scourer might spend some money on solving their problem. The cement manufacturer could well afford to investigate potash manufacture from potash-bearing rocks in connection with cement manufacture.

If potash is to be produced in this country in quantity, some one must spend some money. There has been a noticeable reluctance on the part of American capitalists to finance serious research work, or prospecting by deep core drilling. To begin with, our mining laws and mineral withdrawal possibilities are discouraging. A German potash mine controls several thousand acres of mineral rights. The necessary capital investment would not be justified if this control was not legally obtainable. Potash mines with the necessary concentrating plants are expensive and a large acreage control is necessary. Deep core drilling or other development work on government lands by private individuals would not be justified during the present stage of development of the art of conservation as practiced by our legislators. A positive control of acreage must go with such preliminary work.

There is a large market for potash and the demand is growing. Possibly the lever of necessity may develop sufficient power to overcome the prevailing inertia. Up to now Searles Lake is the only nearby producer on the map.

Discussion

George S. Rice, Pittsburgh, Pa.—I am especially interested in the mining side of this important question. In 1911, when in Europe investigating mining matters, giving special attention to subsidence due to mining, for there is a very big problem of that kind, as many of you know, in the Pennsylvania anthracite region, I visited Stassfurt, which is one of the chief centers of potash mining, and which was pointed out as the place to see surface subsidence. We found the town much askew—some of the streets having sunk from 15 to 30 ft., leaving many ancient buildings pitching at various angles. One old church, 500 or 600 years old, has been sunk about 20 ft., cracking it badly.

I understand that this condition came about because in early mining days they were careless about leaving proper supports and used no filling or packing. The Stassfurt mines work along a buried anticline the sides of which pitch at 45° on the average. Through this careless mining they had a break of the hanging wall which let in water from the overlying marls and sands. The water rapidly dissolved the salts in the pillars, and spreading through the entire chain of mines, caused their complete loss, with serious trouble for the old city. They had to start new mines some distance away, and take care to prevent roof caves.

The packing system is of interest. They were driving large rooms into the rock-salt foot wall and using the rock salt as dry filling.

The application of hydraulic filling, using a saturated solution, was under contemplation at some plant nearby. Whether or not the plan was carried out I do not know. I was reminded of the plan by Mr. MacDowell's remarking on the use of a saturated salt solution in the core drilling.

His reference to the apparent breakdown of the potash syndicate raises an interesting question. We have had the German system of syndicates pointed out as perhaps one means of solving our troubles in the soft-coal mining industry, due to overproduction. It seems to have worked out admirably in the case of the Westphalian coal-mining syndicate and also in the syndicate of Upper Silesia; but apparently the syndicate plan did not work in the potash industry.

One thing that was possibly a helpful factor in the success of the Westphalian syndicate was that the Prussian government decided some years ago, so I understand, to appropriate all future developed extensions of the coal fields to the north of the known field; this probably prevented that bad feature of the American coal industry, the promiscuous opening of mines.

I hope Mr. MacDowell can give us a little light on why the potash syndicate failed.

Charles H. MacDowell.—The question of controlling prices by sales through a syndicate has several sides to it. In Germany, almost since the beginning of the industry, they have tried to restrict the number of new mines. They called in an old coal-mining law which required two shafts—a safety measure. The owners of the mines simply split up the holdings, put down a second shaft, formed a new mining, and asked for additional quotas. The great trouble over there has been to curtail the activities of the man who has mining shares or rights to sell. In order to satisfy the large number of mines they had to put up the price, causing new companies to be floated so they pyramided. Then the new monopoly law made matters worse.

At the time this monopoly law was introduced there were 52 mines,

and now there are 160. The increase came largely from the fact that the people felt the profits must be large under a government monopoly.

I do not think these large selling cartels can be made successful unless there is some method of limiting the number of mines. The business is increased by propaganda; but the increase does not keep pace with the willingness of people to put money in new properties. There is always a fight as to percentages, and one group works against another. Unless there is some way of controlling output, I question whether the syndicate method is a successful one.

It is not wise to mine more than is needed, and if there are too many producers, wasteful mining methods are sure to be resorted to, to cheapen mining costs. Output control and fair prices are necessary if true conservation is desired.

V. C. Grubnau.—Has any process been discovered for obtaining the potash salts from wool scourings?

Charles H. MacDowell.—Yes, a fair tonnage is obtained in Germany and other countries from wool scourings.

Another suggested source of potash is feldspar, concerning which Dr. A. S. Cushman has recently published an article. He estimates the cost of producing muriate of potash from feldspar as $30 per ton. That is a high cost, for the contract we had with the Germans, over which our trouble arose, was based on a price of $19 per ton; and some of the mines, as I have stated, claimed to have made it for $10. Unless war-time conditions continue, I think that muriate of potash from feldspar will not have much chance.

Investigation of Sources of Potash in Texas

BY WILLIAM B. PHILLIPS,* AUSTIN, TEXAS

(New York Meeting, February, 1915)

THE possible sources of potash salts in the United States have been considered from many points of view during the last several years, but it is only within the last two or three months that the situation has become acute. We import from Germany, the only source of large supply, more than $10,000,000 worth of potash salts annually, and this business has been badly crippled by the European war.

If there was ever a time in our industrial existence when the necessity of providing for our own needs was more acute than it is now, we are not aware of it. One has only to study the price lists of chemicals sent out by the large supply houses to see the change wrought within a very short time. Whether or not these prices are to maintain for a considerable time, no one can say; but it appears to be probable that we shall not see prices materially lower for a period of uncomfortable length, to say the least.

It is not my purpose at this time to discuss the various sources of supply of potash salts that have been suggested or recommended. Some of them might afford a pleasing relief but all of them combined would not affect the situation seriously. If we are to have our own sources of potash salts, they will have to compare, in some efficient manner, with the sources upon which we have drawn during the last 20 or 30 years; *i.e.*, they must exist in minable quantities.

It seems to me to be quite idle to speculate upon our domestic sources of potash if those sources are not of a character similar to those upon which we have so long depended. A few thousand or, perhaps, a few hundred thousand dollars' worth of chemically derived potash may be produced under the stress of present conditions, but when we consider that each year demands more than $10,000,000 worth, we must look to underground sources; we must look to mines similar to the famous Stassfurt and Halle deposits. If we cannot mine potash salts, we shall have to do without them for a while.

* Director, Bureau of Economic Geology and Technology, University of Texas.

Is there any general locality within the United States where deep boring for potash salts can be recommended? Is there a working chance anywhere?

This is an important question and one to which a satisfactory reply can hardly be made now.

I venture to mention some circumstances that encourage me to believe that certain parts of Texas are distinctly within the possibilities. I must not be understood as saying that I believe potash salts in working quantities do exist in Texas, but I do believe that the chances are well on the affirmative side. These circumstances are as follows:

In the deep well at Spur, Dickens County, 200 miles northwest of Fort Worth, from a depth of 2,000 to 2,200 ft. was obtained a very salty water which carried 324 grains of potassium chloride per U. S. gallon. The composition of this water was as follows (analysis by S. H. Worrell, Chemist to the Bureau of Economic Geology, University of Texas):

	Grains per U. S. Gallon
Calcium sulphate	1,406.19
Calcium chloride	679.02
Magnesium chloride	219.20
Sodium chloride	3,410.55
Potassium chloride	324.14
Total	6,039.10

In a recent publication,[1] J. A. Udden, who carefully examined the borings from the 4,489-ft. well, and collected samples of water from it, reports on the potash content (as chloride) of 15 samples of water taken from depths ranging from 800 to 3,000 ft. below the surface. The minimum amount of potash in grains per U. S. gallon was 21.4, in drip water from 1,550 ft.; and the maximum amount was 324.1 grains, at a depth of 2,200 ft., the average being 77.7 grains. The table of analyses submitted is given on p. 440.

In commenting on these results, Dr. Udden says:

"The differences in the analyses made here indicate the presence of a potash-bearing stratum somewhere near 2,200 ft. below the surface in this well. The sample having 324.1 grains per U. S. gallon was taken when work on the well ceased for a while, in April, 1912. All the other samples were taken two months later, before the work of drilling was resumed. The water had been standing undisturbed for two months, and the potash-bearing ingredient in the entire seepage had evidently been more evenly distributed through all the water in the hole."

[1] The Deep Boring at Spur, *Bulletin of the Bureau of Economic Geology*, University of Texas, No. 363, p. 83. See also an article by the same author, in the *American Fertilizer*, December, 1912.

Potash Content (as Chloride) in 15 Samples of Water taken at different depths in S. M. Swenson & Son's Deep Boring at Spur, Tex.

A	B	C
800	800	72.2
1,360	300	59.1
1,390	1,390	61.9
1,500 (drip)	1,550	21.4
1,600	1,600	42 9
1,800	1,390	94.4
1,830	1,830	47.2
2,000	2,000	41.0
2,200	2,200	324.1
2,200	1,500	113.3
2,300	2,300	60.2
2,400	2,400	69.6
2,600	1,500	86.2
2,690	2,400	29.1
3,000	1,500	43.6

A. Depth at which sample was taken, in feet below surface.
B. Depth to surface of water, when sample was taken, in feet below surface.
C. Grains of potassium chloride per U. S.-gallon.

Tests for potash in the cuttings were made on 28 samples, representing material taken from depths ranging from 732 to 2,392 ft. From 2,042 to 2,047 ft. there was a trace of potash. The material was a blue shale, effervescing slowly in acid. From 2,068 to 2,110 ft. there was a pronounced trace of potash. The material here was a deep brown, highly ferruginous and slightly micaceous sandy shale, and dark gray, soft anhydrite-bearing dolomite. With these two exceptions, none of the cuttings gave a trace of potash. In commenting on the tests made on the cuttings, Dr. Udden says:

"It can hardly be a mere coincidence that traces of potash were found in the solid rock only near the level where the potash content was highest in the well water. The very unusual amount of 324 grains per gallon also strongly suggests the existence of more than a mere slight impregnation in some stratum. Even after the potash had dissolved from the exposed walls of this stratum and had been diluted through a column of water 3,000 ft. high, it was present in an amount averaging 60 grains per gallon for all the water in the well."

The well at Spur is the deepest well ever bored in Texas. It reached a total depth of 4,489 ft. The elevation at Spur is 2,274 ft. above sea level. The upper 1,250 ft. represent the Red Beds of the Permian, a part of the Double Mountain formation. The next 2,850 ft., of dolomite, anhydrite, sandstone, and shale, are taken to be the equivalent of the Wichita, Albany, Clear Fork, and (in part) the Double Mountain, and possibly are correlated with the Delaware formation west of the Pecos River. The

lower 389 ft., limestone and shale, are thought to correspond to the Cisco (Carboniferous) formation of central Texas.[2]

Three beds of pure salt were found in this boring, five-sixths of it being in the lower half of the Red Beds. One bed, 10 ft. thick, was at 570 to 580 ft.; another, 5 ft. thick, was from 633 to 638 ft., while another, 9 ft. thick, was from 732 to 741 ft. Salt was also noticed at about 1,600 ft. and again at 2,244 to 2,264 ft.

Under the heading "Prospecting for Potash," p. 87 of the *Bulletin* referred to, Dr. Udden remarks:

"Considering the great value of a workable deposit of potash, it seems worth while to call attention to another circumstance in connection with these observations. In either direction, north or south, from Spur, the formations lie practically horizontal for at least a hundred miles, and the potash-bearing formation, whether it be such or not in other places, must be at about the same depth as here, in these directions. It seems to the writer that the general conditions indicated in this boring, the existence of great salt beds and beds of anhydrite, together with the proven potash-bearing stratum, warrant an examination for potash in water from the same horizon in any boring made in this territory. East or west from Spur the depth to this horizon will be greater westward and less eastward. The elevation of the railroad depot at Spur is 2,274 ft. above sea level. This is 666 ft. higher than the elevation at Cisco, about 120 miles to the east-southeast. A line connecting these two points may be taken to follow the direction of the general dip of the formations to the west. The bottom of the well may be taken to represent the beds outcropping at Cisco. On this assumption, the general dip between Cisco and Spur, a distance of 120 miles, will be equal to the depth of the Spur well less the difference in elevation of the two places. This gives us a dip to the west of nearly 32 ft. per mile. Our inability to fix the precise level in the Cisco formation reached in the boring may make this figure either a little too high or too low, but it cannot be far from right. Taking into consideration the general east slope of the land surface, which averages 6 ft. per mile, any stratum should come nearer the surface at the rate of 38 ft. per mile eastward from Spur.

"Assuming now that this general dip is constant between the two points and that the formations are continuous, the horizon which yielded potash in the Spur well should outcrop in a belt where the land surface intersects the dipping plane lying 2,200 ft. below the surface at Spur. This belt would extend through Haskell and Jones counties. . . . Along the line of the Kansas City, Mexico and Orient Railroad, in these counties, the potash-bearing horizon may be looked for at depths of from 100 to 400 ft. It is not to be expected that potash should be found in any outcropping rock in this belt, owing to surface leaching, but well waters might show its former existence."

As one goes west from Spur and crosses Blanco Canyon into Crosby and Lubbock Counties, there are no deep wells from which records are available, nor are any to be had from the counties still farther west, such as Hockley and Cochran. The same is true of the counties to the southwest, such as Lynn, Terry, Yoakum, Gaines, Dawson, Andrews, Martin, etc., lying along or adjacent to the New Mexico border.[3]

Charles L. Baker is now engaged in making a detailed examination of

[2] Udden, *ut supra.*

[3] A recent analysis shows 9 per cent. of soluble potash in material from a well 23 miles northwest of Amarillo, Potter County, at depth from 875 to 925 ft.

the water resources of Hale County, for the Bureau of Economic Geology and Technology, and his report will include the analyses of a large number of well waters from that area, northwest of Dickens County. In the Texas counties immediately southeast of New Mexico there are many old salt basins, half-dried up lakes, etc., which afford not only the usual alkaline compounds, such as salts of lime, magnesia, and soda, but considerable deposits of almost pure sulphate of soda as well. The accompanying photograph (Fig. 1) shows the salt flats at Toyah Lake, Reeves County. It is in such areas that the search for potash salts should be conducted.

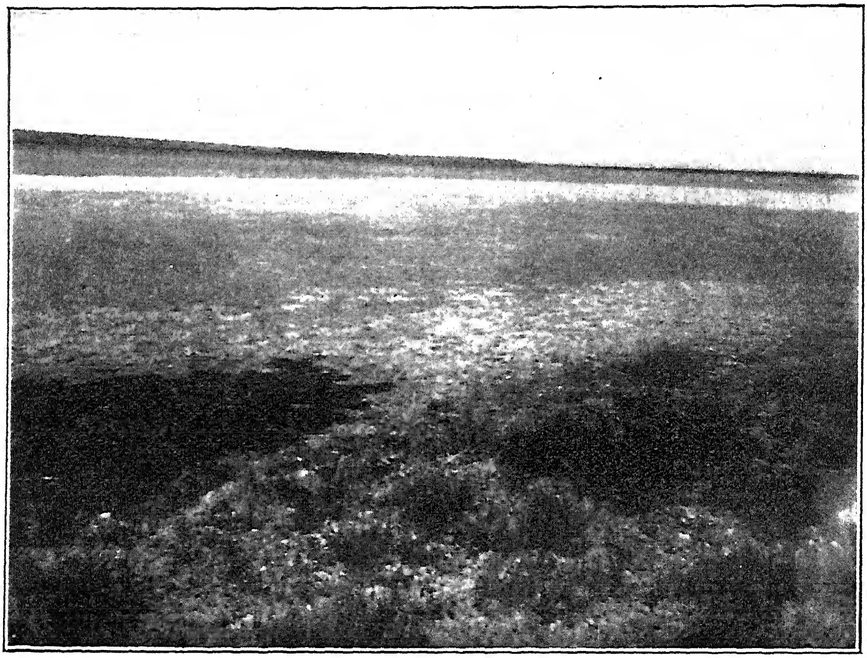

Fig. 1.—Salt Flats at Toyah Lake, Reeves County, Texas.

But boring operations there would involve a large outlay of cash, for it does not appear probable that the cost would be less than $10 a foot. The cost of the 4,489-ft. well at Spur was $50,000, or about $11 a foot, although the salvage reduced the cost to $45,000. Boring operations in the counties less favorably situated with respect to transportation, fuel, water, and supplies are not likely to be conducted for a less cost than the Spur well and in many localities the cost would be greater. No such operations should be undertaken without a thorough geological examination, and when the cost of this is added to that of the boring and of the examination of the cuttings, water, etc., a most important consideration, the undertaking would have to be backed by ample capital. The State of Texas still owns a good deal of land in those counties, as the following

statement, taken from the report of the Commissioner of the General Land Office, Austin, for the two years ending Sept. 1, 1914, will show:

County	Acreage Held by the Public School Fund
Andrews	4,892
Cochran	640
Crane	11,382
Ector	630
Gaines	7,167
Loving	5,420
Terry	1,760
Winkler	4,925
Yoakum	9,736
	46,552

In addition, the University of Texas owns more than 450,000 acres in this part of the State.

The price of this land varies from $2 to $5 an acre for the surface rights. Such lands are sold with reservation of the mineral rights. The mining law passed by the Legislature in 1913 declares it to be the policy of the State not to dispose of the mineral rights in these lands, but to lease them on a royalty basis of 5 per cent. of the gross value of the minerals taken from them. The Public School lands are administered by the Commissioner of the General Land Office, but no provision is made for the examination of such lands with reference to their mineral character. No funds arising from the sale or lease of these lands are, or, under the present law, can be, applied to investigations of a geological or technical nature. Such things are left to private or corporate enterprise. Failing this, there is no source of information available to the public.

Dr. Udden has suggested a search for potash in wells bored to the southeast of Spur, along lines bearing northeast and southwest between the Brazos and Colorado Rivers. In addition to this there should be as detailed an examination as possible in the counties already mentioned, especially in those where extensive deposits of alkali salts are known to exist in old lakes, basins, etc. The more favorable localities for deep boring should be ascertained and mapped, so as to reduce to a minimum the risks attending such operations. If such soluble compounds as common salt, carbonate of soda, sulphate of soda, sulphate of magnesia, etc., are known to exist there, both on the surface and at shallow depths, there is no reason why potash salts, no more soluble than these, should not be found also; *i.e.*, no reason in so far as concerns solubility in water. The average annual rainfall is about 15 in., but so unevenly distributed that for months together there is no precipitation at all. It is during such lengthy droughts that many of the old lakes, basins, etc., appear to be covered with snow, so rapid is the accumulation of salt, etc., due, in part,

to capillarity. Under the winds that blow almost constantly and under a sun that becomes oppressive, at times, evaporation proceeds with extraordinary rapidity. A shallow lake to-day becomes a bed of glistening salt to-morrow.

I have thus far dealt with facts and suggestions which have been ascertained and ventured upon with reference to that part of Texas which lies southeast of and adjacent to New Mexico. There is still another area, bordering upon old Mexico, in the counties of Brewster and Presidio, which has afforded some interesting data with regard to nitrate of soda and nitrate of potash. I have personally examined all of the localities from which these minerals have been reported, and while I cannot say that I have seen anything of commercial value, still, the occurrences are of unusual interest.

In June, 1914, I received from a friend in Alpine, Brewster County, a sample of mineral which was identified as a mixture of nitrate of soda and nitrate of potash. I sent a part of the mineral to Ledoux & Co., New York, and received from them, under date of July 6, 1914, the following analysis:

	Per Cent.
"Moisture	0.77
"In the undried sample:	
Potash (K_2O) soluble in water	30.46
Soda (Na_2O) soluble in water	4.16
Nitrogen (as nitrate)	7.63
Equivalent to nitric anhydride (N_2O_5)	29.42

"The sample also contains small amounts of water-soluble lime, chlorine and sulphate radicle; it is, therefore, impossible to state the exact contents of potassium nitrate and sodium nitrate without making a more complete analysis than the character of the sample warrants. But the data given above are sufficient, assuming sodium to be combined as nitrate, to say that sodium nitrate amounts to 11.4 per cent. and potassium nitrate to 41.50 per cent., leaving a remainder of about 11 per cent. of potash (K_2O) combined as other salts. On the other hand, if all of the nitric anhydride is assumed to be combined with potash, the amount of potassium nitrate would be 55.10 per cent., requiring 25.65 per cent. of potash and leaving a remainder of 4.8 per cent. of potash, together with all of the soda, combined as salts other than nitrate."

Upon receipt of this analysis and after securing further information, I visited the locality in August, and obtained many samples. An average of the better grade of material was sent to Ledoux & Co., and under date of Aug. 28, 1914, they reported as follows:

"The sample of earth containing nitrates of potash and soda marked 'Tinaja on Sec. 120,' referred to in your favor of the 15th instant, has been examined as follows:

	Per Cent.
Water-soluble salts (by solution and evaporation)	29.24
Insoluble sandy matter (by difference)	70.76

On the basis of the original sample, the water-soluble salts contain:

	Per Cent.
Silica	0.10
Calcium	0.46
Sodium	1.00
Potassium	9.71
Chlorine	1.44
Sulphate radical	1.07
Nitrate radical	15.52
	29.30

"These elements and radicles are probably combined as follows:

	Per Cent.
Calcium sulphate	1.52
Sodium chloride	2.37
Sodium nitrate	0.28
Potassium nitrate	24.96
	29.13

"The aqueous solution of the salts contains a trifling amount of organic matter, but it is free from iron, aluminum, or magnesium salts. The composition of the dry salts would, therefore, be practically:

	Per Cent.
Calcium sulphate	5.2
Sodium chloride	8.1
Sodium nitrate	1.0
Potassium nitrate	85.7
	100.0

The question of commercial quantities arises at once. I regret to say that from a close examination of the locality and the results of chemical tests of many samples of the rock in place, I am unable to express a favorable opinion.

The potassium nitrate was found as incrustations on, and thin seams in, a porous sandstone of Cretaceous age. The exact locality is on Sec. 120, Block G-4, Brewster County, Texas.[4] This section is east of Maverick Mountain about 5 miles, and is between this mountain and the more immediate western foot hills of the Chisos Mountains. The nearest railroad point is Marathon, or Alpine, stations of the Southern Pacific Railway, in Brewster County, from 230 to 260 miles southeast of El Paso. The Terlingua quicksilver district is from 6 to 12 miles west.

The sandstone involved dips to the north and east from 7° to 10°, but is exposed only at points where the surface drift, etc., has been worn away by water. Around Chisos Pen, about 2½ miles to the northeast, this

[4] Thin seams of nitrate of potash occur also in a red porphyry in this vicinity.

sandstone carries beds of a sub-bituminous coal which has been used under steam boilers at some of the quicksilver mines.

At the particular place where the nitrate was first found, by a Mexican named Miguel de la O., on Sec. 120, Block G-4, the flood waters of Alamo Creek have cut down into the sandstone and made a fine exposure of 25 to 30 ft. I say the flood waters, for this creek is ordinarily entirely devoid of water except where it has been possible for it to excavate a *tinaja* (water-hole) in the rock. Such a tinaja, when well sheltered, may hold water for several months, or from one rainy season to the next. There is a large tinaja of this kind at this place and fairly good drinking water may be obtained through the greater part of the year. That this circumstance has an important bearing on the accumulation of nitrate of potash is, I think, evident from the local conditions.

The sandstone beds are of unequal hardness and some of the upper ledges now project over the lower ledges by several feet and afford sufficient shelter to be used by Indians. Under one of the more prominent of these overhanging ledges there is now a considerable pile of ashes, half-consumed pieces of charcoal, remains of crude pottery, arrow-heads, etc. This débris carries a good deal of nitrate, but only in the upper part, upon which the nitrate from the roof has fallen. The roof of this half-cave is incrusted with nitrate, some of it hanging loosely to the sandstone and falling at a slight jar. Running through the roof are seams of almost pure nitrate of potash varying in width from 1/8 to 1 in., and these appear also at the back of the cave. But behind the incrustations and alongside of the seams there is a mere trace of potash in the sandstone itself. Two inches behind a heavy incrustation of nitrate the sandstone barely reacts for potash, at 6 in. there is no reaction at all.

After removing the incrustation of nitrate with a chisel the sandstone shows no impregnation with nitrate. The seams of nitrate do not enter the rock, but are confined to the surface or to a plane but slightly below the surface.

I think that the nitrate of potash here has not been derived, by capillary action, from deeper-lying deposits. From what we know of the sources of this material elsewhere, we shall have to look to animal excreta, especially human excreta, for its origin here.

It is, I think, beyond question that this tinaja was a camping ground for a great many years. The water is well sheltered and abundant for the greater part of the year. The adjacent country is even to this day a favorite range for deer, quail, etc., and the dry bed of the creek affords a comparatively easy and safe route to the Rio Grande, for hunting or war parties.

I made a diligent search for other like places in the vicinity, but found none. Many samples of sandstone, shale, etc., were examined for potash, but not one revealed its presence. It is only at this particular tinaja that

any potash has been discovered, and the conditions here seem to me to preclude commercial possibilities, however interesting from a scientific standpoint. Aside from such circumstances as have been mentioned, there is one that militates most effectively against the probability of the existence of workable deposits of potash anywhere near the surface; and that is, the mean annual rainfall in the district. While this cannot be stated accurately, it is not likely to be less than 15 in. This means that upon every square mile there falls every year not less than 285,000,000 gal. of water. During a residence of nearly 18 months in the adjacent quicksilver district, I observed the most awful storms, the eroding, dissolving, and transporting effects of which were most remarkable. The erosion that has taken place at the tinaja and that is now in progress is abundant evidence of the destructive nature of these torrential storms. It is possible that some isolated deposit of nitrates might have been buried under one of the great lava flows which characterize that region, but there is no evidence of such a thing. To base active prospecting upon such a remote possibility is not at all advisable.

The accompanying photograph (Fig. 2) shows the curious effects of erosion at the tinaja and the shut-in character of the tinaja itself. The long well-like excavations in the sandstone are the halves of what were once pot-holes. In places these are 10 to 15 ft. in length and from 4 to 12 in. in diameter. I counted more than 50 of such pot-holes, some of them drilled entirely through an overhanging ledge of sandstone, some of them now in process of drilling, and some drilled for 10 or 15 ft. and then cut through vertically by the water. On the Linville River, western North Carolina, are many beautiful illustrations of pot-holes worn in a sandstone. In a country where the mean annual rainfall is probably not less than 15 in., as in this part of Brewster County, and where the porous nature of the sandstones allows the rapid absorption of water, it would not appear that there has been much, if any, opportunity for the accumulation of workable deposits of such soluble substances as nitrate of soda and nitrate of potash. Some time ago I was called to Presidio County, Texas, to examine a so-called discovery of nitrate of soda. The samples that had been sent in were of an excellent character, running as high as 71 per cent. of nitrate. The locality is about 2½ miles east, a little north, from Candelaria, a little settlement on the Rio Grande, 45 miles south of Valentine, a station on the Southern Pacific Railway.

The nitrate was found as thin veins in a dense, hard trachyte which had been eroded near the top of a small canyon so that cave-like excavations had been formed. A similar occurrence was noted several miles to the west of Candelaria and at this place some old workings were plainly visible. Many samples of the nitrate and of the inclosing rock were examined, but in no case did there seem to be any reason for further prospecting. The thin veins of nitrate are highly localized, and, while rich

specimens were obtained, the commercial possibilities are of no moment.[5] The accompanying photograph (Fig. 3) shows the situation to great advantage.

Several years ago there were reports of the discovery of nitrate of

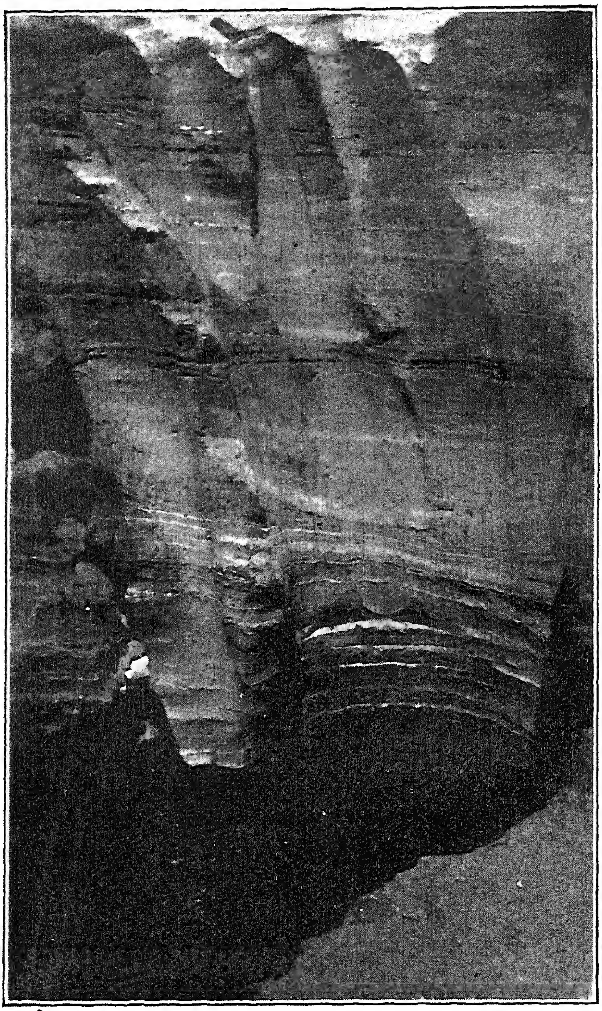

FIG. 2.—EROSION OF SANDSTONE AT TINAJA ON SEC. 120, BLK. G-4, BREWSTER COUNTY, TEXAS, WHERE NITRATE OF POTASH IS FOUND.

potash in El Paso County, but investigations showed that while the mineral did exist in almost pure condition, the amount was extremely small.

[5] See articles by the writer in the *Engineering and Mining Journal*, vol. xc, No. 27, p. 1303 (Dec. 31, 1910); *Manufacturers' Record*, Jan. 19, 1910, and Aug. 11, 1911; the *Mexican Mining Journal*, vol. xiii, No. 3, p. 21 (Sept., 1911), and the *Mineral Industry*, vol. xix, p. 616 (1910).

From time to time, during the last 10 years, I have examined many localities in west Texas from which specimens of nitrate of soda and nitrate of potash were obtained, but I have yet to see a place which presents commercial possibilities of any moment whatsoever. I make this statement at this time to allay certain reports which have found some credence among those whose hopefulness has run away with their judgment. Rich specimens have been found at more than one locality, but in every case the situation has been such as to forbid any expectation of the finding of workable deposits.

Mention should be made of a locality on the Devil's River, Val Verde

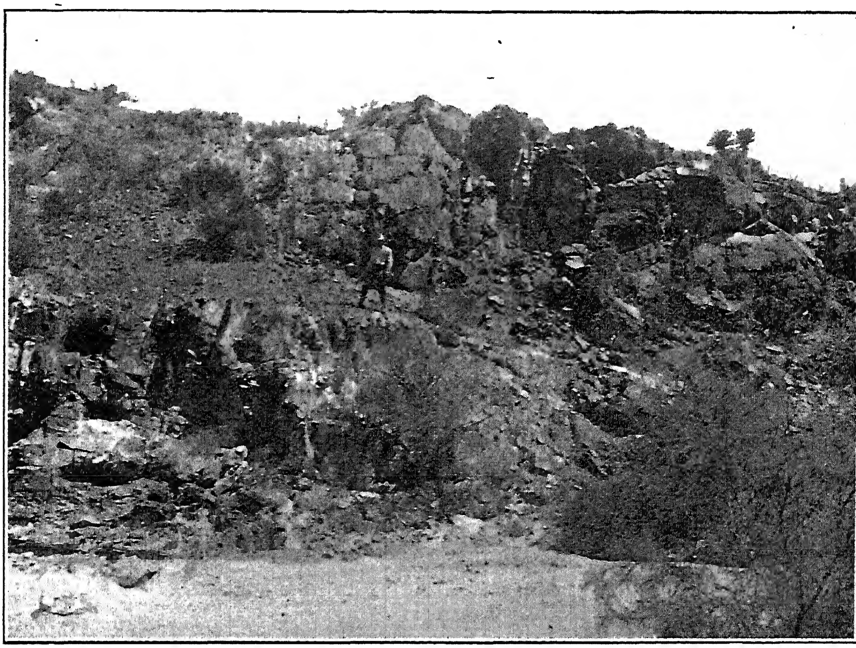

Fig. 3.—Exposure of Trachyte Carrying Thin Seams of Nitrate of Soda, near Candelaria, Presidio County, Texas.

County, described by J. A. Udden,[6] which presents many of the characteristic features of so-called "deposits" of nitrates in Texas.

The saltpeter here was found in a cave on Devil's River and was formed by natural process from human excrements and blood waste. The place was an old camping ground and the accumulated rubbish, 136 ft. long and about 34 ft. wide, averages 5 ft. in thickness. In the uppermost foot there was 20 per cent. of saltpeter, in the second foot 7 per cent., in the third

[6] Report on a geological survey of lands belonging to the New York & Texas Land Company, Ltd., in the Upper Rio Grande Embayment in Texas (Augustana Book Concern, Rock Island, Ill., 1907).

foot 1 per cent., in the fourth and fifth feet less than 1 per cent., while the sixth foot contained none.

The cave is located under an overhanging cliff of limestone about $\frac{5}{8}$ mile from where Indian Creek empties into the Devil's River. A little potash is found in certain bat caves in Texas, which have been partly burned, the potash appearing in the ashes, but the amount present is commercially negligible. The largest of these caves, containing, probably, several thousand tons of bat guano, is in Burnet County, about 25 miles northwest of the town of Burnet, and about 14 miles west of the railroad from Burnet to Lampasas.[7]

In summing up the entire matter, it can be said that the only hopeful outlook for the existence of workable beds of potash salts in Texas is in the direction already indicated by Dr. Udden, and in the almost wholly unknown region southeast of and bordering on New Mexico. Explorations in either of these areas would have to be in depth, with systematic chemical examination of cuttings and return waters.

[7] Phillips, William B.: The Bat Guano Caves in Texas, *Mines and Minerals*, vol. xxi, No. 10, p. 440 (May, 1901).

The Plasticity of Clay and Its Relation to Mode of Origin

BY N. B. DAVIS,* ITHACA, N. Y.

(New York Meeting, February, 1915)

OUTLINE

I. Introduction

While working with a number of very sticky cracking clays from western Canada the writer became interested in a study of the cause of the excessive plasticity. This led to a review of the rather bulky literature on the subject and the finding that plasticity still remains to be satisfactorily explained. Most of these investigations seem to have been carried on solely from the chemical viewpoint, the geologic aspects having been ignored. In nearly every case, the investigators have worked with the higher-grade clays of low to medium plasticity, such as the kaolins, the excessively plastic ones having generally been neglected, partly because of their limited application, yet the author believes that these are

* Cornell University.

the clays through which the problem should be attacked along with a study of the formation of clay deposits.

II. Definition of Plasticity

Numerous definitions of plasticity have been advanced from time to time by different writers and most important have been those proposed by Seger, in Europe, and Ries in America.

Seger[1] describes plasticity as "that property which enables a solid to receive a liquid into its pores, to retain the latter completely, thus enabling the mass, by pressure and kneading, to take any desired shape, to retain the shape unchanged on the removal of pressure and extraction of the liquid and the consequent change to the solid state. It is not found in all friable solids. Many cannot be molded when mixed with a liquid or on its removal will not possess sufficient cohesion of the particles to result in a strong body."

Other materials such as wax, tar, hot glass, etc., are described as plastic, but plasticity in this sense must not be confused with that in clays. Plasticity in the case of lead or hot glass is really malleability.

In explaining plasticity in the case of clay we have only to deal with that property developed on the addition of water and the subsequent drying to a body of some strength. Increased hardening and shrinkage usually accompany increase in plasticity.

III. Theories of Plasticity

The various theories that have been advanced from time to time to explain the plastic properties of clay may be classified as follows:

A. Structure of the clay particles.
   (1) Fineness of grain.
   (2) Plate structure.
   (3) Interlocking particles.
   (4) Sponge structure of particles.
B. Presence of hydrous aluminum silicates.
C. Molecular attraction between particles.
D. Presence of colloidal gelatinous matter.

In the following pages the writer will attempt a review of the various theories which it seems to him have tended to lead up to the colloidal one.

A. Structure of the Clay Particles

In 1867 Johnson and Blake,[2] working on kaolinite, tried to explain plasticity as due to the fineness of grain, and plate structure of the kaolinite and other platy minerals present in clay.

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[1] *Collected Writings*, vol. i, p. 533.

[2] *American Journal of Science and Arts*, 2d ser., vol. xliii, p. 357 (1867).

Clays in which they found bundles of kaolinite crystals, seemed to be of lower plasticity than those in which the bundles were broken up into their component cleavage plates. They give two analyses of kaolins of practically the same chemical composition, yet one is "fat" and the other is "short." From this they infer that the difference in the degree of plasticity is due to the difference in the state of division.

Since the publication of this view, similar explanations have been advanced by such writers as Biedermann and Hersfeld,[3] and Cook[4] in 1878, and more recently by Haworth and Wheeler,[5] Atterberg,[6] and Le Chatelier.[7]

Experimenting with fine grinding of minerals developing a plate structure, Wheeler reported that calcite and gypsum finely ground in a ball mill with water developed plasticity; furthermore, test pieces gave tensile strengths ranging from 100 to 350 lb. per square inch. Talc and pyrophyllite also showed some plasticity when treated in the same way, but the tensile strength was low. It is altogether probable that the high tensile strength reported in the case of calcite and gypsum was due to the fact that some of the material went into solution, and crystallized out as a cement on drying. This did not happen with the other two minerals, which are more resistant to solution.

Recently a clay was received at Cornell University by Dr. Ries which analyzed 98 per cent. dolomite, and under the microscope was seen to be made up of little rhombs about 0.008 mm. in diameter. It was as plastic as a kaolin and of low tensile strength.

Orton[8] tried the effect of fine grinding on glass and noted a slight plasticity but no tensile strength.

R. T. Stull[9] ground mica and found a "surprising increase in plasticity." He concluded that fineness of grain and plate structure were the cause of plasticity.

Grimsley and Grout[10] ground samples of quartz, calcite, and mica, and noted a development of some plasticity in each case.

Precipitated barium sulphate mixed with 14 to 22 per cent. water was described as plastic,[11] but the material showed little tensile strength on drying. Minerals that split into scales or plates, such as barite, kaolinite, muscovite and other micaceous minerals, were described as

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[3] Bischof: *Die Feuerfesten Thone*, p. 23.

[4] *Report on Clay Products*, New Jersey Geological Survey (1878).

[5] *Clay Deposits*, Missouri Geological Survey, vol. xi (1896).

[6] *Zeitschrift für angewandte Chemie*, vol. xxiv, No. 20 (May 19, 1911).

[7] *van Bemmelen's Gedenkboek.*

[8] *Brick*, vol. xiv, p. 216.

[9] *Transactions of the American Ceramic Society*, vol. iv, p. 257 (1902).

[10] *Clays, Limestones and Cements*, West Virginia Geological Survey, vol. iii, p. 46 (1905).

[11] Atterberg: *Loc. cit.*, p. 928.

strongly plastic if ground and elutriated so that the size of the particles did not exceed 0.002 mm. diameter. Materials like quartz, feldspar, and gypsum do not cleave so readily into plates and hence are not so plastic. Atterberg concluded that the plasticity of the European clays is due to the presence of mica meal in those of the north, and to kaolinite scales in those of the south.

Another German writer[12] believed that the particles were of extreme fineness and long or round in shape. The long thin particles gave greater contact surface, thereby increasing surface tension and plasticity.

It is evident that the conceptions of what constitutes a plastic mass, held by the writers cited are at variance, except in the case of mica. Wheeler, and Grimsley and Grout, describe quartz, calcite, and gypsum as plastic, while Atterberg says they are not.

Rohland[13] mentions the fact that clays ground in a drag mill are more plastic than those ground in a ball mill; because in the former the particles are brought to a smooth flat form, whereas in the latter the grinding produces angular or irregular forms. He adds that even if artificial means produced a structure similar to that developed in natural processes it would not develop plasticity in the clay but simply represent one of the factors favoring plasticity. Again, Leppla[14] attributed plasticity to the small size of the crystal plates, to inflexibility without elasticity, and to the excellent cleavage and softness. Further, he was of the opinion that the adhesion between water and the crystal grains was stronger than the cohesion of the smallest cleavage pieces.

Two Russian investigators[15] came to the conclusion that plasticity is due to the interlocking of the clay particles and depends for its best development on a proper mixture of very fine and coarse grains. At an earlier date, Olchewsky[16] suggested the idea of interlocking hook-like particles, and Termier,[17] studying a series of French residual clays and shales, found numerous little hook-like aggregates to which he also attributed the binding power. A study of this by Wheeler in a number of Missouri clays showed, that in all the clays examined in which the hooks were found, the plasticity was low. By grinding to break up these hook-like crystals of kaolinite into their cleavage plates, plasticity was increased.

Daubrée[18] found that long-continued grinding of feldspar developed some plasticity, and Olchewsky explained this as due to the porous spongy

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[12] Linder: *Tonindustrie-Zeitung*, vol. xxxvi, No. 27, p. 382 (Mar. 2, 1912).

[13] *Die Tone* (Wien, 1909).

[14] *Baumaterialien Kunde*, vol. ix, p. 124 (1904).

[15] Aleksiejew and Cremiatschenski: *Zap. imp. russk. techn. obschtsch,* xxx (1896).

[16] *Deutsche Topfei-und Zeigler-Zeitung*, No. 29 (1882).

[17] *Comptes Rendus de l'Académie des Sciences*, vol. cviii, p. 1071 (1889).

[18] A. S. Cushman: *Transactions of the American Ceramic Society*, vol. vi, p. 66 (1904).

structure of the finest particles caused by the removal of the alkalies in solution. Of all the earlier writers Olchewsky came closest to a true explanation of plasticity.

B. Presence of Hydrous Aluminum Silicates.

To fineness of grain and plate structure Vogt,[19] Arons,[20] Bischof,[21] Seger,[22] and others have added the idea of the presence of hydrated aluminum silicates in clay as a cause of plasticity. It was noted that in general the temperature at which kaolinite lost its water of constitution was coincident with the disappearance of plasticity. This was on the assumption that kaolinite formed the basis of all clays—an assumption we now know to be incorrect.

Recent experimenters[23] have shown that all the combined water of the clay is not part of the clay base in the sense of water of hydration. Clay is a mixture of minerals, crystalloids and colloids, and hence has no definite dehydration temperature, water being lost more or less continuously. The clay substance of the impure clays was found to represent a type essentially different from that of the purer materials. Dehydration does not necessarily completely destroy the plasticity and hence "the combined water appears to have no direct connection with the phenomenon of plasticity." The conclusion that dehydration does not destroy the plasticity is questioned by Rohland,[24] who contends that it is lost at the temperature at which the combined water begins to be evolved, 590° to 620° C. It is probable that these authors have not the same conception of what constitutes a plastic mass or where plasticity ends. It is probable that some of the clays used by Brown and Montgomery contained considerable plate-like particles which made the material weakly plastic.

C. Molecular Attraction between Particles.

Grout[25] was the first to develop this theory to any extent in connection with clays, although it was hinted at by Ladd,[26] Beyer,[27] and Leppla[28] before he took it up. He describes plasticity as due to the following factors:

1. The distance the clay particles can move on each other without

[19] Ries: *Clays, Their Occurrence, Properties and Uses*, 2d ed., p. 47 (1908).

[20] Dammer: *Chemische Technologie der Neuzeit*, vol. i (1910).

[21] *Die Feuerfesten Thone.*

[22] *Collected Writings*, p. 69.

[23] Brown and Montgomery: Dehydration of Clays, *Technical Paper No.* 21, *U. S. Bureau of Standards* (Apr. 25, 1913).

[24] *Silicat Zeitung*, vol. ii, p. 30 (1914).

[25] *Clays, Limestones and Cements*, West Virginia Geological Survey, vol. iii, p. 54 (1905).

[26] *Bulletin No.* 6a, *Georgia Geological Survey* (1898).

[27] *Annual Report, Iowa Geological Survey*, p. 88 (1904).

[28] *Loc. cit.*, p. 124.

losing coherence, which varies with (a) shape and size of grain, (b) distance through the water film the particles will attract each other.

2. The amount of the coherence or resistance to movement, which varies with (a) the friction in the films, (b) the friction of the grains on one another.

He writes that fineness of grain, plate structure, and colloids are insufficient to explain plasticity and proposes a theory based on cohesion and adhesion, suggesting that the high plasticity of some clays is due to the greater attraction of the grains of those clays for water. Such an attraction depends on the constitution of the molecule and any change in the chemical constitution would afford a reasonable explanation of the fact that clays of similar chemical composition but of different molecular arrangement vary in plasticity.

Grout's molecular attraction comes down to nothing more than capillary attraction caused by the water in the interstitial spaces between the clay particles. It is well known that water wets different substances (minerals) to different degrees. We can add a small amount of water to a mass of dry ground spar, quartz, or calcite, and in each case the mass will hold together better than when dry simply because of the capillary force exerted by the water between the grains. When the mass is dried out it will fall down to a powder again simply because the capillary force no longer acts. If other material, such as a suitable colloid gel, is added another factor is introduced and the mass does not disintegrate. This brings the discussion up to the last theory.

D. The Presence of Colloidal Gelatinous Matter

Most writers at the present day believe that colloidal matter is the cause of plasticity, but our ideas of the colloidal state have been rather confused with the rapid advance of colloid chemistry and consequently the conceptions of colloids held by the various writers have differed somewhat, especially as regards clays.

In taking up this last theory it would perhaps be best to lead up to it through a discussion of the present-day conception of the colloidal state of matter.

(1) *Review of the Theory of Colloids.*—In the early '60's, Thomas Graham,[29] an English physicist, working on the phenomenon of diffusion of dissolved substances through organic membranes, found that certain substances passed through freely, while others did not do so, or at most, very slowly. In general he found that substances of pronounced crystal habit passed through membranes; while other substances, amorphous in character, such as gelatin, gum arabic, etc., apparently in solution, did not diffuse.

This led him to divide matter into two great classes, colloids and

[29] *Philosophical Transactions of the Royal Society*, vol. cli, pp. 183 to 224 (1861).

crystalloids. Further work along this line showed that substances, formerly considered soluble, could be obtained in what appeared to be true solution, yet they would not diffuse. These apparent solutions he called colloidal, or "sols."

He found that slight additions of electrolytes, either acids or bases, which did not react chemically with the substance in colloidal solution, tended to precipitate or coagulate it. These coagulated sols he called "gels" because of their jelly-like nature.

Graham worked with colloidal solutions of silicic acid, tungstic acid, chromium, aluminum, and ferric hydroxides. He was not the first investigator to prepare these sols, but he was the first to attempt a systematic investigation of their properties.

Since Graham's time great advances have been made in the study of the colloidal condition of matter, and colloid chemistry has developed as a branch of physical chemistry.

The term colloidal refers to a physical condition that may be taken on to a greater or lesser degree by all substances. Colloid chemistry is the study of the properties of substances in the finely divided state.

In considering the colloidal condition we have to think of two phases being present, namely the solvent and the colloidally dissolved substance. Ostwald[30] in Europe, and E. E. Free[31] in America, in papers published about the same time have classified the possible phases as follows, calling the solvent the dispersive phase and the colloidally dissolved substance the disperse phase. Either phase may be solid, liquid, or gas.

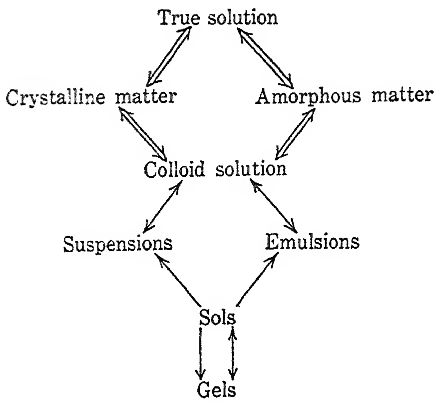
Dispersive Medium	Disperse Phase		
	Gaseous	Liquid	Solid
Gaseous	Class 9: G + G. No real example as gases always are completely miscible.	Class 8: G + L. Fog *e.g.*, at the point of liquefaction of gases, atmospheric fog, etc.	Class 7: G + S. Smoke, cooled ammonium chloride, vapor, etc.
Liquid	Class 6: L + G. Foams.	Class 5: L + L. Emulsions.	Class 4: L + S. Suspensions; gels.
Solid	Class 3: S + G. Gas inclusions in minerals (pumice), or in metals.	Class 2: S + L. Liquid inclusions in minerals; occluded water, water of crystallization.	Class 1: S + S. Inclusions of solid particles in minerals; solid solution.

G = Gas L = Liquid S = Solid

[30] *Kolloidchemische Beihefte*, vol. ii, pp. 94 to 97 (1910).

[31] *Journal of the Franklin Institute*, vol. clxix, p. 424 (1910).

Most important are the liquid-solid and liquid-liquid systems, represented by suspensions and emulsions respectively. Any substance, amorphous or crystalline, may take on the colloidal condition, and there is a tendency toward an equilibrium between these three states and true solution. This may be represented diagrammatically as follows:



A large number of sols of the metals and their oxides, hydroxides, and sulphides are known and a study of them is becoming more and more important in explaining the formation of ore deposits. Many of them are present in clays and affect their properties to different degrees.

Certain substances, such as the gums, gelatin, agar, etc., are known only in the amorphous and colloid states, while other substances, such as the metallic oxides, etc., mentioned above, are known in all the different states. Sodium chloride is a substance we commonly know as crystalline and in true solution, yet under suitable conditions it can be prepared as a colloid.

There are a large number of organic compounds occurring in nature which can be dissolved as sols without any special precautions. As an evidence of their condition as sols, and not in true solution, it has been found that they have little or no effect on the boiling or freezing points of the solvents.

By the use of the ultra-microscope, invented in 1905 by R. Zigmondy and H. Siedentopf, we have been able to observe the size of the particles in colloidal solutions. The particles of many sols have been measured and shown to be larger than the calculated size of the molecules. This, along with the inability to diffuse, and the non-effect on the boiling and freezing points, are the fundamental differences between colloidal and true solutions.

As Ostwald[31] has stated, viscosity is another important property

[31] *Transactions of the Faraday Society*, vol. ix, pp. 34 to 46 (1913). *Chemical Abstracts*, vol. viii, p. 848 (Mar. 10, 1914).

of fundamental importance in investigating the properties of the colloidal state, on account of the extreme sensitiveness of this constant toward very slight alterations in the conditions of colloid. The metal and sulphide sols have been found to have little effect on the viscosity of the solvent while organic sols have been found to show marked increase in viscosity. The organic sols are usually liquid (emulsions) and this has led us to consider that in colloidal solutions showing an increased viscosity the disperse phase is a liquid.

(*a*) Suspension Colloids (Suspensoids).—It was shown by Stokes in 1850, and since has been verified by others, that a small round particle falling in a liquid assumes a constant velocity; and as the radius of the particle becomes smaller, the rate of falling decreases. Hence, with very small particles, we may have a suspension that will appear stable for a long time. If the particles are irregular in shape, as they usually are, the velocity of fall is slower, due to increased resistance.

Besides gravity, other factors are to be considered in examining the stability. The most important, perhaps, is the electric charge taken on by the particles when suspended in the liquid medium. Clay particles suspended in water take on negative charges, and hence continually repel one another. This repulsion may be one cause of another phenomenon known as the Brownian movements, so named after their discoverer, Dr. Brown, an English botanist. The movement may be noticed in all colloidal suspensions. The particles are seen to be jumping around in all directions in the medium, yet not touching one another. This movement tends to keep the particles in suspension.

On the addition of many foreign substances, sometimes in very small amounts, changes in the suspension are noted. The organic colloids and strong alkalies tend to increase the stability and are known as protective colloids and deflocculators respectively. The neutral salts and acids cause flocculation and are known as flocculators. Much work has been done on the phenomenon of flocculation and deflocculation; and for a more detailed treatment of the subject the reader is referred to the work of E. E. Free,[32] and to that of H. E. Ashley.[33]

In 1874 Schloesing[34] applied the theory of colloid suspension to explain the action of clay suspensions on the addition of electrolytes. He allowed clay suspensions to settle for 27 days, and then evaporated the solution with the material remaining in suspension. The sediment from this liquid dried to a horny mass, and this he called colloidal.

Since Schloesing's time, P. Rohland in Europe, and H. E. Ashley in America, have been most active in experimenting along this line.

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[32] *Journal of the Franklin Institute*, vol. clxix, pp. 421 to 438 (1910).

[33] Technical Control of the Colloidal Matter of Clays, *Technical Paper No.* 23 *U. S. Bureau of Standards* (November, 1911).

[34] *Comptes Rendus de l'Académie des Sciences*, vol. lxxix, pp. 376 and 473 (1874).

Ashley's work has been most important because of his attempt to control the colloid content of clays.

(*b*) Emulsion Colloids (Emulsoids).—It has been known for some time that the addition of small amounts of colloids belonging to the emulsion class greatly increases the stability of the suspension. Those that do this are given the special name of protective colloids. They are much less sensitive to electrolytes than the suspensions, and their effect is explained by assuming that each particle of the suspension is coated with a thin layer of the emulsion colloid, and then takes the electric charge of the latter. This action is important in explaining the plasticity of clay and will be referred to again.

The most important inorganic emulsoid is silicic acid. On the addition of hydrochloric acid to a sol of this acid it will set to a gel and the change is not reversible. It was thought at first that this gel represented a definite hydrate, but the work of Ramsay, Spring, van Bemmelen and others on dehydration has shown that it is not a definite hydrate. There is a continuous loss and gain of water as the humidity of the surrounding atmosphere varies. Similar work on the hydrates of iron, aluminum, chromium, etc., has shown that compounds of this nature are not definite hydrates, but in the same way take on and lose water with the varying conditions.

Löwenstein[35] experimenting with clays found that they lost water continuously. He found that as the temperature was raised gradually, there were no breaks in the dehydration curve to indicate definite hydrates of similar dehydration points in the clayey mixture. Reference has already been made to the work of Brown and Montgomery along this line.

Compounds of the silicic acid type have the important property of adsorption of other colloids and salts from solution; and because of this are known as adsorptive colloids. Such hydrous gel bodies are shown to be present in clays by their degree of absorption. Ashley's method of determining the colloid content of clay is based on this adsorptive phenomenon.

Rohland[36] claims to have been the first to demonstrate colloids as a cause of plasticity, but credit should be given Schloesing,[37] whose work remained hidden for so long.

Rohland has written much on the subject; he has dĕveloped partially the colloid theory of plasticity on the basis of the work of Graham and Schloesing.

In his first paper, published in 1902,[38] he pointed out that colloids,

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[35] *Zeitschrift für anorganische Chemie*, vol. lxiii, No. 2, pp. 69 and 139 (July 23, 1909). *Chemical Abstracts*, vol. iv, p. 887 (Jan.–July, 1910).

[36] *Int. Mitt. für Bodenkunde*, vol. iii, p. 492 (1913).

[37] *Comptes Rendus de l'Académie des Sciences*, vol. lxxix, pp. 376 and 473 (1874).

[38] *Zeitschrift für anorganische Chemie*, vol. xxxi, No 1, p. 158 (May 16, 1902).

as distinguished from crystalloids, are substances which possess shrinkage and cohesion on drying, properties associated with plasticity in clays. About the same time Ries[39] published his observations on plastic clays under the microscope and noted that the extremely minute spherical structureless particles might be of the nature of colloids.

In 1903, van der Bellen[40] accepted the theory of the presence of colloids, and states that the purest clays and kaolins are undoubtedly mixtures of aluminum salts of meta-silicic acid, the varying degree of plasticity depending on the presence of a colloid modification in the structure of the clay particles.

In 1904, Ries[41] again writes that "it may seem doubtful whether mere interlocking alone is sufficient to account for the tenacity of the clay, and whether or not organic substances, included under the term of colloids and which are no doubt present in many clays, do not exert some cementing action."

The same year Cushman[42] writes that clays are to be considered as aggregations of particles which fall under three heads: (1) non-plastic crystalline, (2) non-plastic amorphous, and (3) plastic colloid. He states that plasticity and binding power will not only vary with relative proportions of the above-mentioned materials but also with the particular character and activity of the sort of colloid matter that happens to be present. Undoubtedly the principal substance of which all good clays are composed is the hydrated silicate of aluminum, or "kaolin,"[43] and yet the variation in binding power and plasticity is very great.

In his summary he presents the following points:

1. That both plasticity and binding power are merely manifestations of a colloid modification of matter which exists in rocks and clays.

2. That the activity of these useful qualities depends upon the characteristics of the special colloids that may be present, as well as upon their past history, and the modifying effect upon them of saline and organic solution.

3. That absorptive qualities of clays, such as their ability to stick when touched with the tongue, and as exhibited by their occasional use as "lakes," clarifying agents, etc., is to be ascribed to the same cause.

He adds further that the size and shape of the clay grains have yet to be considered in explaining plasticity.

The viscosity of the water films on clay grains[44] suggests that increased

[39] *The Clays and Clay Industry of New Jersey*, New Jersey Geological Survey, vol. vi, p. 83 (1904).

[40] *Chemiker-Zeitung*, vol. xxvii, No. 36, p. 433 (May 6, 1903).

[41] *Transactions of the American Ceramic Society*, vol. vi, p. 82 (1904).

[42] *Idem*, p. 72.

[43] Probably kaolinite is meant.

[44] Grimsley and Grout: *West Virginia Geological Survey*, vol. iii, p. 51 (1905).

viscosity may be due to the presence of "colloids." A plastic clay may be as one composed of finely divided plates or scales held together by mutual attraction and water film tension, aided by colloids if present.

In 1909, Rohland[45] summarized the theories of plasticity and sums up his own work as follows: "My investigations on the decomposition of clays in relation to their plasticity can briefly be summarized as follows:

"Those substances which form colloidal solutions in water develop more or less plasticity. Clay and porcelain bodies contain colloids of both organic and inorganic nature to a certain extent latent, the colloidal condition only being developed in water. The plasticity can be increased by both organic and inorganic colloids."

Soluble salts also seem to have an effect on the plasticity of a clay.[46] In general they are thought to increase plasticity. This has been discussed by Purdy and Moore,[47] who considered it an exceedingly probable assumption that it is the influence of the adsorbed salts that gives to a clay its plasticity. The theory of the cause of plasticity in clays which appeared most tenable to them was that of adsorbed, or otherwise held, salts, resulting in an enveloping liquid media of high surface tension.[48]

Writing in 1912, Grout and Poppe[49] summarized their paper on plasticity as follows. "We find that the essential peculiarity of a plastic clay is the fact that the water used in wetting the clay is somehow rendered viscous. In explanation we have two well-known phenomena: molecular attraction and the action of colloids. Both are certainly present. Both are necessarily active. The calculation indicates that molecular attraction may be quantitatively sufficient, especially since the force is known to be variable and may be greater in clays than in any other mineral. Colloids alone would probably be quantitatively insufficient. No experiments have shown them capable of such effects in other mineral powders."

These authors have made the mistake of calling clay a mineral when in reality it is a mixture of minerals. The writer believes that colloid gelatinous films are quantitatively sufficient, when their relation to the solid clay particles in natural clays is properly understood.

Had Grout used the microscope more he would have noticed that clay particles are not clean-cut bodies, but, as will be shown later, are coated with gelatinous material, the inner edge of which is intergrown with the more rigid part (see Fig. 1). On adding gel material to a clean crystalline or amorphous powder or granular mass, the coatings have not

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[45] *Die Tone* (1909).

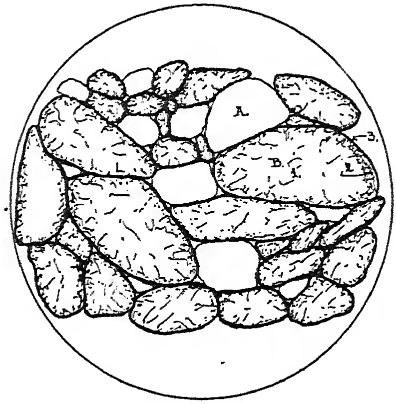
[46] Seger: *Collected Writings.*

[47] *Transactions of the American Ceramic Society,* vol. xix, p. 222 (1907).

[48] *Idem,* vol. xi, p. 582 (1909).

[49] *Idem,* vol. xiv, p. 80 (1912).

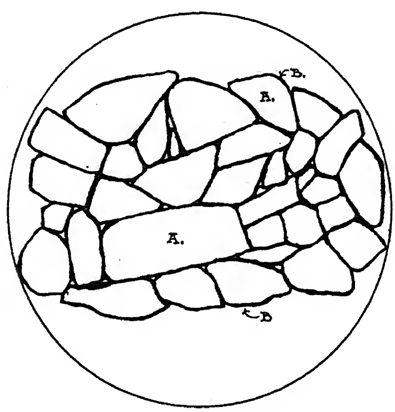
this intimate connection with the particles and hence such a mixture has not the same strength (see Fig. 2).



A. Crystalline unaltered grains (quartz).
B. Crystalline grains undergoing decomposition.
1. Interior of grains showing decomposition cracks.
2. Zone of more intense decomposition with the formation of new, more stable, crystalline compounds and gelatinous modifications.
3. Gelatinous envelope.

FIG. 1.—NATURAL CLAY.

In recent papers Atterberg[50] and Rohland[51] both consider colloid gels as a cause of plasticity in some degree. Atterberg lays stress on the



A. Fresh undecomposed grains of a dry-ground powder.
B. Coating of artificial colloid gel material on the grains (gelatinous silica, alumina, etc.).

FIG. 2.—ARTIFICIAL MIXTURE.

size and shape of grain, while Rohland holds that this has nothing to

[50] *Int. Mit. für Bodenkunde,* vol. iii, No. 4 (1913).
[51] *Idem,* vol. iii, No. 6 (1913).

do with plasticity but that colloids are *the* cause. Without a doubt both are effective but the colloidal gels are most important.

If we are to understand that all very finely divided matter is in the colloidal state, then all clays contain colloidal material; hence the size of grain has something to do with plasticity, and the best plasticity is to be found in mixtures of colloidal and non-colloidal matter.

The chemical composition and physical properties of different substances in the colloidal gel state vary. For example, we may take a number of glues, which are colloidal substances, and note that the adhesive properties of each vary somewhat. In like manner the adhesive and cohesive properties of the various sticky colloidal substances in clays vary.

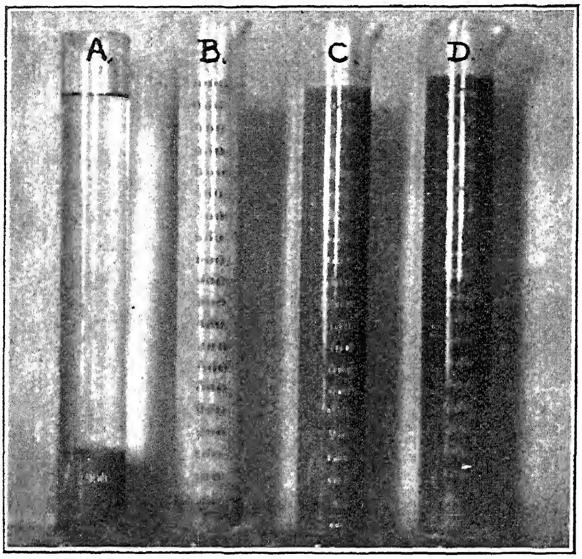


FIG. 3.—FOUR CRACKING CLAYS AFTER 20 DAYS' SETTLING.

(2) *Experiments.*—As the writer stated in the introduction to this paper, he believes that the best way to study plasticity is to work with excessively plastic clays. Most of the work in the past has been done with the higher-grade clays, such as the kaolins, fire clays, ball clays, etc. Perhaps because of their few uses the excessively plastic cracking clays have been neglected.

A beginning was made on four excessively plastic clays from western Canada, 50-g. samples being taken and suspended in distilled water in 100-c.c. measuring tubes. After allowing the suspension to stand for 24 hr., samples were taken by means of a pipette at different depths, and the size of the largest particles in each case examined was measured under the microscope. One clay, on firing, showed considerable soluble

salts and the suspension settled in a few hours (see A, Fig. 3). The others remained in suspension for 20 days, when the particles were measured again. This time the particles were so small in A and B that they could not be seen with a magnification of 600 diameters. However, the particles in C were visible, and measured 0.002 mm. in diameter. The sizes of the particles in the three clays remaining in suspension are shown in the curves of Fig. 4; and Fig. 3 shows the clays as they appeared in suspension at the end of 20 days.

A study of the curves shows that something is present in D causing larger particles to stay in suspension than in C and B. Also the particles remaining in suspension in C are larger than those in B. The pronounced effect of the "something" in D is shown by the size of the particles remaining in suspension after 20 days' settling.

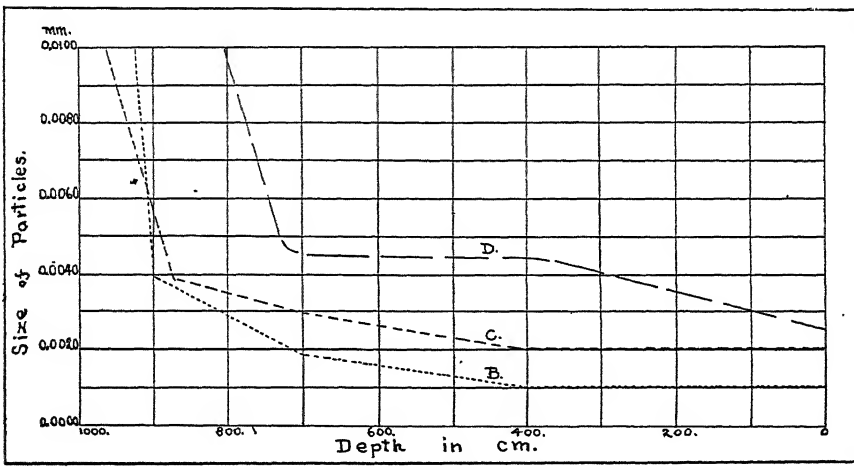


FIG. 4.—SETTLING CURVES OF SUSPENSIONS SHOWN IN FIG. 3.

This difference in the suspensions suggested a possible difference in the nature of the colloidal material present, so analyses for colloidal silica and organic matter were made. The relative depth of color in B, C, and D suggested that the percentage of colloidal organic matter increased toward D.

The following are the partial analyses:[52]

	B	C	D
Colloidal SiO_2	3.66	0.76	0.12
Carbonaceous matter (mostly colloidal)	0.21	0.26	1.10

In making the determination of colloidal silica in D the sodium carbonate extraction was dark in color, due to the colloidal organic matter,

[52] Published by permission of the Director of the Geological Survey of Canada.

while in C it was much cleared, and in B comparatively clear. The effect of the large amount of colloidal organic matter in D was shown by the fact that, after 20 days, the particles in suspension could still be seen under the microscope. On the other hand it was most striking to find the colloidal silica percentage increasing toward B.

As stated in the discussion of colloids in general, the emulsoid colloids raise the viscosity of solvent. And as shown by Grout,[53] organic colloid gels have a much greater effect on plasticity than the inorganic colloid gels. This is well shown in the above analyses.

The organic colloid material may have the effect, as an emulsoid, of increasing the viscosity of the suspension, or may act as a protective colloid coating on the clay particles.

In these three clays plasticity is attributed to the colloidal silica in B, to the mixture of colloidal silica and organic colloids in C, and to the large proportion of organic colloids in D; without a doubt, other factors are working toward plasticity. Samples C and D both burn to a red, while B burns to a cream color, hence we may expect some colloidal iron oxide gel in C and D; and gelatinous alumina may be present in all three. However, organic matter and silica are most important.

According to Cameron and Bell,[54] little or no aluminum hydroxide forms in ordinary rock weathering. They examined several thousand soils from all parts of the United States and found aluminum hydroxide in only one sample, which came from California.

Recently an attempt was made to prove the presence of hydrous alumina in clays by recalculating analyses.[55] Such a method is of value in recalculating fresh rock analyses, but it cannot be applied to clays. Kaolinite was assumed as the basis for these calculations, it being considered the basis of all clays, an assumption we know to be incorrect.

The work of Bleininger and Brown[56] on the viscosity of clay slips is of interest in this connection. They determined the relative viscosity of a number of clay slips, and found that the results roughly arranged the clays in the order of their plasticity. It would have been interesting to have had analyses for colloidal silica, organic colloidal matter, etc., on the clays used by them.

To test the effect of organic matter on a suspension, an attempt was made to duplicate natural conditions by putting into a ball mill a couple of pounds of Delaware washed kaolin, water, and an ounce of peat, with a couple of balls to stir. After mixing for 10 hr., the slip was drawn and

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[53] *Clays, Limestones and Cements,* West Virginia Geological Survey, vol. iii, p. 80 (1905).

[54] *Bulletin No.* 30, *the U. S. Bureau of Soils,* p. 24 (1905).

[55] M. G. Edwards: The Occurrence of Aluminum Hydrates in Clays, Discussion by Ries, *Economic Geology,* vol. ix, No. 4 (June, 1914).

[56] *Transactions of the American Ceramic Society,* vol. xi, p. 604 (1909).

allowed to dry out. It was then mixed with water to a plastic mass. Comparing the resultant plasticity with that in a portion of untreated kaolin, it was found to have increased. Samples of the treated and untreated material were then suspended in water and allowed to settle. After 10 hr. it was observed that the untreated kaolin had settled clear, while the treated portion showed considerable matter in suspension. Test pieces showed that the shrinkage had increased from 4 to 8 per cent. There was no appreciable change in the tensile strength.

Another experiment was tried by mixing a ferric sulphate solution into powdered feldspar, and then precipitating gelatinous iron oxide around the grains by adding ammonia. The material showed some plasticity and the tensile strength was raised slightly.

Many similar experiments have been tried in the past by various investigators. Mention may be made of the work of Zimmer[57] in adding gelatinous silica to samples of kaolin. He found both plasticity and binding power increased. Grimsley and Grout[58] added 0.08 per cent. agar to two clays and found the plasticity increased. A similar experiment using gelatinous alumina required a larger amount (3 per cent.) of this gel to raise the plasticity as high as did the 0.08 per cent. agar. On drying in air, and wetting again, they found that the sample containing the alumina gel had dropped back to the original plasticity. From this they concluded that since plastic clays are not so injured by air drying, it is evident that such colloids do not explain plasticity. Mixtures of gelatinous alumina and silica acted in the same way. In these experiments the investigators failed in this way, because they were not duplicating natural conditions. Had they precipitated their gelatinous material in the mass and hence got it in intimate contact in thin films over the grains they would have found the plasticity to remain about constant.

As Ashley[59] has pointed out, it has been the custom for many years to add gelatinous matter to clay to improve its working qualities. Gelatin, glue, starch, gum arabic, dextrin, milk, and silicate of soda are gelatinous substances often added to pottery bodies. In India, cow dung, horse dung, rotten paper, gum, and starch are added to the local clays in making pottery.

Acheson's[60] patent to improve plasticity consisted of adding a colloid, tannin, to clay; and Auclair's[61] patent consisted of dissolving out the colloid gels with alkali, and removing the sol by filter pressing, the

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[57] *Transactions of the American Ceramic Society*, vol. iii, p. 36 (1901).

[58] *Loc. cit.*, p. 48.

[59] Technical Control of the Colloidal Matter of Clays, *Technical Paper No.* 23, *U. S. Bureau of Standards*, p. 82 (November, 1911).

[60] British Patent No. 7776, Apr. 3, 1907.

[61] French Patent No. 372858, Dec. 22, 1906.

material remaining being pure crystalline ceramic paste. Keppler[62] patented a process to improve plasticity by deflocculating with alkali, adding colloidal humus matter, and reprecipitating the whole with acid.

It is well known that boiling a clay with water for even a short time decreases the content of crystalline quartz and kaolinite, and increases the proportion of colloidal matter.[63]

Hilgard[64] states that boiling serves to reduce the kaolinite ingredient of soils and clays to the amorphous, plastic, diffusible modification. By boiling a very slightly chalky pipe-clay for 85 hr. he succeeded in rendering it quite plastic. He was simply making conditions favorable for the mineral matter to go into solution.

In the experiments in which colloidal gel material was added to kaolin and feldspar, the plasticity was increased but the tensile strength was little affected, at least not to the extent expected when comparing the chemical composition of natural clay mixtures and artificial mixtures. This may be explained by a more careful study of Olchewsky's suggestion of a porous felty structure of the grains.

The examination of clay particles under the microscope shows that they are not crystalline all through. They show alteration to secondary products (crystalline, amorphous, and colloid), especially around the outside and following in on cleavage and other cracks.

All inorganic crystalline and amorphous substances are soluble to some extent in water, and this solubility is increased by the presence of salts or acids.

It is well known that the solubility in water of any substance is never zero, although it may be exceedingly small. Consequently kaolinite in contact with water dissolves to some slight extent and is immediately hydrolyzed. The solution thereby becomes unsaturated with respect to kaolinite; consequently more dissolves, which in turn is hydrolyzed, when still more can be dissolved, and so on. These two simultaneous processes may thus transform an appreciable amount of kaolinite, provided sufficient time be allowed—a condition which would usually obtain in nature.

This point of view is in harmony with conclusions reached by H. Gedroiz,[66] who asserted that aluminum hydroxide, ferric hydroxide, silicic acid, and kaolinite are, at the moment of their formation by weath-

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[62] *Chemical Abstracts*, vol. iii, p. 494 (Jan.–July, 1909). German Patent No. 201987, Aug. 4, 1906.

[63] Ashley: *Loc. cit.*, p. 84.

[64] *American Journal of Science and Arts*, 3d ser., vol. xvii, p. 211 (1879). Ref. to by Ashley, *loc. cit.*, p. 84.

[65] Ashley: *Technical Paper No.* 23, *U. S. Bureau of Standards*, p. 36 (November, 1911).

[66] *J. Exp. Landw.*, pp. 272 to 293 (1908).

ering, hydrosols, which stay as such or are coagulated, depending on the composition of the soil solution.

Mellor[67] found that the outlines of the solid particles of wet ground feldspar could be more readily stained with dyes than before the water treatment, indicating some change on the surface of the grains.

Of the common rock-forming minerals Buckman[68] gives the following list ranging from quartz, the least soluble, to nepheline, the most easily attacked:

1. Quartz
2. Muscovite
3. Biotite
4. Orthoclase
5. Plagioclase
6. Epidote
7. Serpentine
8. Talc
9. Hornblende
10. Augite
11. Apatite
12. Olivine
13. Leucite
14. Nepheline

In the light of colloid chemistry we may consider minerals, or any mineral for that matter, in contact with water, tending to go into solution. Certain constituents will go to form new more stable crystalline compounds, and these, taking on the colloidal gel condition with ease, may develop in that state.

In a recent paper on the origin of laterite, A. Luz[69] has tabulated some of the possible changes as follows:

Original Mg-Fe-Ca-K and Na holding crystalline aluminum silicates

Step I. Coll. aluminum silicates	Coll. iron hydroxide (mostly mixed with SiO_2)	Coll. Mg-Ca-K-Na-silicates
Step II. Coll. alumina Coll. SiO_2		Coll. SiO_2 Mg-Ca-K-Na-hydroxides
Step III. Cryst. hydrargillite	Cryst. limonite	Refer to Step I.
Step IV. Cryst. diaspore (Possible but not observed)		

Fig. 1 represents a number of mineral grains very much enlarged, the interiors of which are criss-crossed by lines of alteration while the outer rims show, first, a zone of solution; second, a zone of colloid gels and new crystalline material; and third, a water film. In this diagram the writer sees an explanation of the impossibility of duplicating the degree of plasticity and binding power of clays by making up mixtures of fresh crystalline and colloid gel matter in the laboratory. There is not the same intimate connection between the gelatinous matter and the crystal-

[67] *Transactions of the English Ceramic Society,* vol. v, p. 72.
[68] *Transactions of the American Ceramic Society,* vol. xiii, p. 346 (1911).
[69] *Kolloid Zeit.,* p. 86 (1914).

line skeleton as in colloid and crystalline mixtures in nature. We have not been able to duplicate the time element as yet.

Ashley[70] describes an experiment in which he shook up portions of Georgia sagger clay in 100-c.c. graduated tubes, with water to make 100 c.c. After shaking for an hour, they were allowed to stand for some time and then most of the clear liquid was siphoned off to be replaced by fresh water. This was repeated about 80 times and it was found that the bulk of the sediment had increased. He found this in agreement with the observations that the sediment becomes bulkier on prolonged contact with water, and indicates that the amount of colloidal material increases under these conditions. This is exactly what happens in nature when a residual kaolin of low plasticity is worked over by water, transported, and deposited as a sediment. This will be taken up in more detail under the formation of clays.

IV. The Formation of Clays

In taking up the study of the formation of clays as an aid in explaining plasticity it would perhaps be best to follow some genetic classification. The author recognizes how hard it is to get a complete classification to cover all possible cases. As this is intended more as a general study of the question it would perhaps be best to use as simple a classification as possible. That suggested by Ries[71] comes nearest to this and with minor changes will be used, as follows.:

- A. Residual clays.
  - 1. Kaolins or china clays.
    - (a) Veins derived from pegmatite.
    - (b) Blanket deposits.
  - 2. Buff- and red-burning residuals, derived from different kinds of rock, igneous, metamorphic, or sedimentary.
- B. Transported clays.
  - 1. Deposited in water.
    - (a) In running water.
      - (1) Flood-plain clays, usually impure and sandy.
      - (2) Glacial stream clays in drift and always stony (boulder clays).
    - (b) In still water.
      - (1) Lacustrine or lake clays.
      - (2) Estuarine clays.
      - (3) Marine clays.
  - 2. Wind-formed deposits, loess.

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[70] *Technical Paper No.* 23, *U. S. Bureau of Standards*, p. 48 (November, 1911).
[71] *Clays, Their Occurrence, Properties and Uses*, 2d ed., pp. 27, 28 (1908).

A. *Residual Clays*

Residual clays may be defined as those clays resulting from the weathering of rock in place. Rock may be igneous, sedimentary, or metamorphic.

In countries that have suffered glaciation, clays of this type are rarely found, but in those parts of the world that have escaped glacial action, or where erosion is not too fast, residual clays are common. In general, then, most residual clays are found in warm countries.

The processes of weathering, working toward the formation of this type of clay, may be classified as follows:

1. Mechanical or disintegration.
   (a) Changes in temperature.
   (b) Work of plants and animals.
2. Chemical or decomposition.
   (a) Solution.
   (b) Hydration.
   (c) Oxidation.
   (d) Deoxidation.
   (e) Recombination.

As a result of these processes acting on rocks of all kinds, residual clays are formed which may be classified under two sub-heads, depending on their uses or burning qualities. Those resulting from the disintegration and decomposition of granitic and gneissic rocks high in the feldspathic minerals, and low in the iron-magnesium minerals, are the high-grade kaolins or china clays. Those resulting by the same process acting on more basic igneous and metamorphic, as well as on the impure sedimentary rocks, are the lower-grade clays, ranging from those used for making fire brick to the poorest common-brick clays.

Kaolins.—The theories advanced to explain the formation of kaolin from feldspathic rocks may be classified as follows:[72] (a) simple weathering, (b) post-volcanic emanations, (c) ascending spring waters, containing CO_2, (d) waters draining from swamps or peat bogs, (e) sulphuric acid solutions, (f) by the alteration of sericite.

There are examples of kaolin deposits formed by all the processes, but the most common is that by simple weathering.

Kaolins formed by all the above methods are of low plasticity, and it is lowest in those in which water played a minor rôle, as in post-volcanic emanations.[73]

The chemical reactions that go on are not definitely known in any case, and are extremely complicated. Little more than a shrewd

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[72] Ries: *Transactions of the American Ceramic Society*, vol. xiii, p. 51 (1911).
[73] Jackson and Richardson: *Transactions of the English Ceramic Society*, vol. ii, p. 59.

guess can be made, based on a knowledge of the original minerals and some of the residual products.

By whatever process a kaolin deposit may have been formed, it is generally conceded that feldspar was the principal original mineral. The alteration in feldspar has been shown by Cameron[74] and others to be essentially one of hydration, and may be represented as follows:

$KAlSi_3O_8 + HOH = KOH + HAlSi_3O_8$
(orthoclase)

$HAlSi_3O_8 - SiO_2 = HAlSi_2O_6$
(pyrophyllite)

$HAlSi_2O_6 - SiO_2 = HAlSiO_4$
(kaolinite)

$HAlSiO_4 - SiO_2 = HAlO_2$
(diaspore)(?)

The silica may be taken away largely in the sol state, or may remain as a gel, due to coagulating agents being present. It may crystallize out, in part, and appear as quartz.

The alkali may be taken away in solution, or may enter into the formation of muscovite.

Because of the small amount of colloidal gel matter formed, the plasticity is low. The mineral grains are more crystalline than in any other type of clay.

Due to infiltration, organic colloid matter may be present in small amount. It is never very much, except perhaps near the surface of the deposit, where, as Ries[75] has noted, greater plasticity is developed. This is not to say that organic colloidal matter is the sole cause of the increase in plasticity at the surface, but it is one cause.

Kaolins do not vary much in their chemical composition and physical properties, as shown by the table on the following page, compiled from the report by Watts.[76] Yet some of them do seem to have present a substance which increases the tensile strength and the air shrinkage at the same time, properties increased in effect by the presence of gels.

The plasticity of residual kaolins is low, and may be attributed to a mixture of fine and coarse plates or particles, and minute amounts of gelatinous silica, alumina, organic colloids, etc.

Buff- and Red-Burning Residuals.—By far the most commonly occurring residuals are the buff- and red-burning clays, derived from all kinds of rocks, except those from which kaolin develops. Passing in igneous rocks, from granites through gabbros to the most basic periodotites, the mineral composition changes from feldspar-mica-quartz, through

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[74] *Bulletin No.* 30, *U. S. Bureau of Soils.*

[75] *Maryland Geological Survey*, vol. iv, p. 463 (1902).

[76] Mining and Treatment of Feldspar and Kaolin in the Southern Appalachian Region, *Bulletin No.* 53, *U. S. Bureau of Mines*, p. 89 (1913).

Analyses and Tests on Washed Kaolins[a]

	SiO_2	Al_2O_3	Fe_2O_3	TiO_2	CaO	MgO	BaO	Na_2O	K_2O	H_2O	Total	Per Cent. Shr. at 110° C.	Cone II Fire Shr.	Ten. Str.	Refr. Value	Total Shr.
I. Kinsland......	50.64	35.57	0.25	0.03	Tr.	Tr.	0.07	0.08	1.70	11.90	100.24	4.40	9.80	8.0	1670°	14.2
II. Buchanan....	46.30	39.06	0.20	0.04	Tr.	Tr.	Tr.	0.11	0.60	13.77	100.08	5.40	13.90	28.5	1730+	19.3
III. Forest Hill..	49.20	37.58	0.17	Tr.	Tr.	Tr.	Tr.	0.13	0.47	12.53	100.08	4.00	9.70	16.0	1730	13.7
IV. Dillsboro....	46.95	37.73	0.15	0.05	Tr.	Tr.	Tr.	0.18	0.60	13.99	99.65	6.10	12.70	24.0	1730	18.8
V. Piedmont.....	48.50	37.35	0.85	Tr.	Tr.	Tr.	Tr.	0.32	1.02	12.00	100.04	4.40	8.10	16.5	1710	12.5
VI. Gurney......	44.00	40.79	0.11	Tr.	Tr.	Tr.	Tr.	0.07	0.55	14.72	100.24	5.40	11.90	27.5	1730+	17.3
VII. McGuire....	46.35	39.00	0.30	Tr.	Tr.	Tr.	Tr.	......	0.50	14.00	100.15	5.70	10.50	27.5	1730+	18.2
VIII. Raby....	46.90	38.60	0.25	Tr.	Tr.	Tr.	Tr.	0.26	0.39	13.80	100.20	6.25	13.50	21.5	1730+	19.75
IX. Smith.......	48.50	37.69	0.31	Tr.	Tr.	Tr.	Tr.	0.02	0.91	12.55	99.53	6.80	12.00	20.5	1670	18.8
X. Southern.....	46.67	39.07	0.11	0.02	Tr.	Tr.	Tr.	0.11	0.25	13.22	99.45	....	....	....	....	....
XI. West........	48.92	36.37	0.37	0.02	Tr.	Tr.	Tr.	0.11	0.29	12.70	98 78	7.00	18.00	24.0	1730	25.0
XII. Sprucepine..	45.20	38.45	0.45	Tr.	Tr.	Tr.	Tr.	......	0.65	14.80	99.55	4.00	12.60	8.0	1730	16.6
XIII. Tolley....	46.35	38.80	0.25	Tr.	Tr.	Tr.	0.03	......	0.41	14.00	99.84	5.40	10.90	8.0	1730	16.3
XIV. Bryson...	46.95	37.24	0.40	0.05	Tr.	Tr.	0.00	0.24	0.49	14.10	99.47	4.00	12.80	14.0	1730	16.8

[a] From *Bulletin No. 53, U. S. Bureau of Mines.*

feldspar-amphibole-pyroxene to amphibole-pyroxene-olivine, with other accessory minerals such as the micas present in lesser amounts. This means a loss of alkali-aluminum silicates, and an increase in iron-magnesium silicates, which are known to be less resistant.[77]

The same may be said for metamorphic rocks, except the feldspar gneisses, which yield kaolins. The more basic hornblende gneisses, slates, schists, etc., all come under this general statement.

The sedimentary rocks vary much in mineral composition and often the mineral particles are so small as to make it impossible to determine their character. Resulting as they have, mainly from water transportation, the colloidal condition is already developed to some degree, and its extent will depend on the age and past history of the deposit.

The limestones yield products ranging from fire clays to the poorest common-brick clays. Sandstones seldom yield clays, while shales usually do, since they are simply hardened clays. Here we are on the border between residual and transported clays, and shales may be included under both. In brick making, shales are often simply ground up and by this artificial means reduced to a soft clay again. However, it is well known that weathering gives a much more plastic material, due to the rejuvenation of the colloid gels and formation of fresh gels.

That mineral composition of the parent rock has much to do with the resultant plasticity in the clay may be shown by the following quotation:[78] "Clays obtained from gabbros and similar basic or dark-colored rocks are usually highly plastic, often deeply ferruginous and in many cases fine grained." In the same report a ferruginous clay of this type is described from near Catonsville, Md. It is a clay of great plasticity and wonderful tensile strength. Granites in the same general area yield plastic ferruginous clays, but they are less plastic, and less ferruginous usually, than those derived from the more basic rocks.

Residual clays in the State of Missouri are largely derived from limestones and shales.[79] The limestone-derived clays are yellow to brown or red in color, are excessively plastic, and consequently can seldom be used because of cracking in drying and burning.

Many other examples of plastic clays of the residual type might be given to show that it is in those clays high in the constituents iron oxides, silica, alumina, etc., commonly assuming the colloidal gel form, that plasticity is best developed.

B. *Transported Clays*

By far the greatest number of clays used in the ceramic industries are the transported clays. They are more widely distributed than the

[77] Buckman: *Transactions of the American Ceramic Society*, vol. xiii, p. 347 (1911).
[78] Ries: *Maryland Geological Survey*, vol. iv, p. 463 (1902).
[79] H. A. Wheeler: *Clay Deposits, Missouri Geological Survey*, vol. xi (1896).

residual type and vary in purity from sedimentary kaolins through ball clays, stoneware clays, fire clays to common-brick clays and the generally useless gumbos. They represent mixtures of the decomposition products of rock of all kinds carried by water in streams and rivers and deposited on flood plains, in glacial drift, in lakes and swamps, in estuaries, and under marine conditions.

The materials making up these clays not only suffer the action of water and other solvent agents, but also the attrition of particle on particle while being worked over by the stream.

It is interesting here to quote from the observations of George H. Cook[80] published in 1878. He was reporting on the clays of the State of New Jersey and had in mind the transported type so common in that State. "Some clays appear to consist of well-defined crystalline forms; others show a few of these in a mass of fragmentary shapes; others still seem to be wholly made up of irregular forms and exceedingly fine particles of matter. A satisfactory explanation of these different conditions is that the more finely divided clays are those which have had their crystalline forms broken up, either wholly or in part by the several agents that have moved them from the place of their origin to their present location, while those in which these forms still abound have not suffered the same constitutional derangement. Now, it has been observed that the former class of clays are more plastic than the latter. And a further observation is that by breaking up these crystalline forms and rendering them finer, the plasticity was promoted."

A very important fact connected with the origin of clay, and one which has great influence upon the plasticity, is that the farther clayey material is transported, during which it is constantly exposed to the abrading and comminuting action of the particles upon themselves as well as upon the banks and bottoms of streams over which they are carried, the finer it is ground and the smaller the scales into which kaolinite particles are broken.[81] The smaller the crystalline plates of kaolinite are divided the more plastic is the clay, so those clays that have been subjected to severe wearing action are likely to be extremely plastic, whereas those that have had little or no wear, as those formed *in situ*, are but slightly plastic. This is well shown in the glacial and loess clays, where mechanical disintegration and abrasion of the clay particles have been great. Such clays are very plastic, though possessing a very moderate amount of kaolinite; while many of the china and flint clays which have not been transported possess almost no plasticity.

(*a*) Clays Deposited in Running Water.—Upstream on flood-plain flats are formed deposits of material carried by the stream from its

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[80] *Clay Deposits of New Jersey*, Geological Survey of New Jersey, p. 287 (1878).
[81] Wheeler: *Clay Deposits*, Missouri Geological Survey, vol. xi, p. 24 (1896).

source. This material may be made up of the wash of all kinds of rocks or only from certain kinds, depending on the geology of the drainage basin. In areas where transportation is fast and the original rock is granitic, the most of the material will be sandy, granular in character, and on the flood plains the deposits will be mostly of a sandy nature with some clayey material. Such a deposit would show little or no plasticity. The feldspathic minerals and quartz are least readily attacked by solution and hence would remain nearly intact. However, where the more basic iron-magnesium minerals are present, the resultant flood-plain deposit shows more plasticity. In either case had the original rock suffered residual weathering before transportation the chances for the formation of colloidal gel products would have been increased.

In the State of Florida a clay occurs that is probably of this type. It is the well-known Florida ball clay. It is supposed to have resulted from the transportation and deposition of material from a feldspar-granite area. It consists of a mixture of white clay and well-rounded quartz pebbles, the latter forming 65 to 75 per cent. of the entire mass.[82] From this natural clay, the fine material is washed out and put on the market as Florida ball clay or kaolin.

In chemical composition the washed product is almost exactly that of the washed kaolins from residual deposits. The ball clay is highly plastic while the residual kaolins are of low plasticity. The only other difference between them is the method of their formation; the attrition and extra water action the ball clay was subjected to in transportation. The explanation of the difference in plasticity, then, must lie in the action of water on the grains.

The character of the clay depends largely upon the nature of the rock formations over which the stream and its tributaries flow.[83] As a concrete example, the alluvial clays along the large streams in the Piedmont Plateau contain a large percentage of quartz and mica sand and other undecomposed minerals derived from igneous rocks, and are simply residual kaolin made plastic by an increase in gelatinous matter in the processes of transportation and deposition.

As a rule flood-plain deposits are of lower grade and are usually used for common brick. Such deposits are made up of material from more basic rocks than are described above. A good example is that occurring near Edmonton, Alberta.[84] It consists of alternating layers of sandy, silty clay, and occasional pockets of gravel. It is used for common soft-mud and dry-press face brick. The run of bank is plastic, gritty, and calcareous. It burns to a light red at cone 03. The material was de-

[82] Ries: *Clays, Their Occurrence, Properties and Uses*, 2d ed., p. 334 (1908).

[83] Otto Veatch: *Bulletin No.* 13, *Georgia Geological Survey*, p. 29 (1909).

[84] Ries: Clay and Shale Deposits of the Western Provinces, *Memoir No. 24-E*, *Geological Survey of Canada*, p. 37 (1912).

rived from the areas of sedimentary rocks drained by the Saskatchewan River.

The flood-plain deposits along the largest rivers in the northern part of Missouri are very plastic, sticky, black clays and are known as "gumbos."[85] Physically they are very fine grained, extremely plastic and tenacious, cracking badly in drying. They show very high tensile strengths, ranging from 270 to 410 lb. per square inch. They have a low specific gravity, 1.98 to 2.05, and contain from 2 to 5 per cent. organic matter. In these clays the organic matter in gelatinous form is the cause of the excessive plasticity.

Yet another example may be taken from Ries's[86] report on Texas clays. This gives analyses of two clays from Harrisburg, Tex. They form different beds in the same bank, and although their chemical composition is practically the same their physical properties vary widely.

	I	II
SiO_2	80.39	80.84
Al_2O_3	9.82	8.09
Fe_2O_3	2.88	2.25
CaO	0.42	1.44
MgO	0.45	0.26
Na_2O	0.19	0.10
K_2O	Tr.	Tr.
TiO_2	0.35	0.78
H_2O	3.11	6.00
	97.61	99.76
Water required for mixing, per cent.	18.7	19.8
Average tensile strength, lb. per sq. in.	188	275
Air shrinkage	4.8	8.6
Plasticity	fair	high
Drying	no cracks	cracks
Absorption cone 5	15.64	8.19
Steelhard	cone 9	cone 5

"A more interesting contrast could hardly be desired, and it forms no exceptions." Can we attribute the high plasticity, high shrinkage and the tensile strength in example II to plate structure, molecular attraction, or mere fineness of grain? Most certainly not. The presence of colloidal gel coatings on the mineral grains is the only logical explanation. Another point is significant. As Zimmer[87] has pointed out, pottery mixtures containing silica in the gelatinous form fuse at lower temperature than mixtures containing crystalline silica or quartz. A similar effect is shown in the burning of these two clays. Clay II becomes steel-

[85] Wheeler: *Clay Deposits*, Missouri Geological Survey, vol. xi, p. 542 (1896).

[86] *Clays, Their Occurrence, Properties and Uses*, 2d ed., p. 64 (1908). *Bulletin No.* 102, *University of Texas*, p. 224 (1908).

[87] *Transactions of the American Ceramic Society*, vol. iii, p. 25 (1901).

hard at a lower temperature than I. Also the absorption at cone 5 is lowest in the case of clay II. Hence it is fair to argue that the difference in burning qualities is due to the presence of more colloidal gel matter in II than I. Organic matter should have been determined and also colloidal silica.

Glacial clays are not often used because of their limited extent and stony character. They are often called boulder clays. The fine material has often been described as "ground rock flour," but that phrase is misleading. In its formation it was ground wet and subjected to water action for some time, a proper condition for the formation of the colloidal state.

As in other transported clays, the material has been gathered from various kinds of rock. The clay formed by material from basic rocks is more plastic than that from granite areas.

(*b*) Clays Deposited in Still Water.—Clay deposits formed in lakes and swamps form an intermediate group between the river and deep-water clays. They are often of limited extent and basin shaped, and again may be quite extensive. Frequently they consist of alternating clay and sand beds. They are common in glaciated regions and are often of glacial origin.

Large areas of Canada are covered by deposits of this type. The early glacial lakes covering large parts of Alberta, between the front of the Keewatin ice sheet and the Rockies, supplied the silty soil worked around Edmonton.

The clays of the glacial lake, Agassiz, cover wide areas in Manitoba, as do these clays of Lake Ojibwa in northern Ontario.

In northern New Jersey similar clays occur, having been deposited in ponds by streams flowing from the front of the retreating ice sheet. They are often fine grained and laminated, and of good plasticity.

The impure plastic clays in swamps along the larger streams and near the coast in the Coastal Plain of Georgia belong to this division of clays. They are very fine grained, plastic and contain organic matter.

In southern New Jersey also occur large deposits of estuarine clays. They were laid down in sheltered bays and estuaries. They are fine grained and interbanded with sand layers.

The extensively worked clays of the Hudson Valley are of this type, the section usually being made up of an upper sand bed, a yellow weathered clay, and a blue clay. The clays are laminated, plastic and red burning.

All these clay types are simply mixtures of mineral matter, crystalline and colloidal gel, and their plastic properties depend on the relative amounts and specific properties of these two. They have all suffered water action for long periods of time.

By far the most important variety of transported clay is that which was deposited under marine conditions. The marine clays are made

up of the finest products of rock decay and transportation. The material was brought down to the ocean by the streams and deposited in quiet water, the finest material farthest from shore. Such formations cover wide areas and are known to be often of great thickness. They occur widespread in beds of Silurian, Devonian, and Carboniferous age. Those of the Mesozoic are also quite persistent.

Many marine clays, since their formation, have been covered by other materials and often are consolidated enough to be called shales. It is well known that where consolidation is due to pressure alone, the plastic properties are little affected, but where there has been a change in the mineral matter and recrystallization has gone on, the plasticity is lowered. Weathering or fine grinding and pugging has then to be resorted to in using the material in a wet or stiff-mud process.

Marine clays vary in mineral and chemical composition and in physcal properties, as do the clays of other types, depending on the composition of the rocks of the drainage basins from which the material was derived.

In the Coastal Plain areas of the Atlantic States occur plastic kaolins and fire clays derived from the residual clays of the highly feldspathic crystalline rocks of the Piedmont Plateau. They were transported only short distances, yet are much more plastic than the residuals from which they were derived.

Other examples could be given of clays, mixtures from different drainage basins, and from basins of basic rocks, but the increase in plasticity in the case of the transported kaolins is most striking.

Wind-Formed Deposits.—In the Mississippi Valley region are found extensive deposits of a silty, often calcareous, clay called loess. It is much used for making common wet-mud and dry-press brick.

The loess clays are of post-glacial age and represent collections of wind-blown material. In Missouri the deposits are of great thickness.[88] They are usually non-stratified and exhibit a columnar structure in the banks. The material is usually fine grained, the particles varying in size from 0.1 mm. to ultra-microscopic dimensions. Mixed with water it develops low plasticity and is often hard to dry safely.

In the light of the reasoning in this paper, low plasticity should be expected in wind-formed clay. As in the other types, the degree of plasticity depends on the past history of the material. The large quantities of calcareous drift gathered by the glaciers from the sedimentary rocks to the north, and surrounding the Mississippi Valley, furnished the material of the loess. The only period during which it was subjected to water action was that of the life of the ice sheet. Following its deposition as drift in front of the retreating glacier, strong winds worked

[88] Wheeler: *Clay Deposits*, Missouri Geological Survey, vol. xi, p. 483 (1896).

it over, carrying away the finest particles to form the loess. Surficial weathering has since taken place to some extent and has increased the plasticity locally.

V. Summary

In review, the following points may be emphasized:

(1) Clean crystalline rock powders do not develop plasticity on the addition of water, no matter how fine they may be ground in the dry form.

(2) Such powders if ground in water for a long time do develop some plasticity.

(3) Those powders containing minerals least easily attacked by water, such as quartz and feldspar, develop least plasticity.

(4) Water exerts a solvent action on the mineral grains, resulting in colloid and true solution, the colloid modifications remaining as coagulated sols, gelatinous coatings over the mineral grains.

(5) Organic gel bodies in small amounts increase the viscosity of the water films in mineral powders. The effect seems to be more pronounced than that of equal amounts of the inorganic gels, as shown by Grout's experiments with agar and with gelatinous alumina.

(6) The amount of gel matter present in a clay depends on the past history of the deposit. Clays are simply the weathering products of crystalline rocks in which the most soluble constituents have been removed, leaving behind more resistant secondary products (crystalline and amorphous) with gelatinous coatings.

(7) The most highly plastic clays are those that have been subjected to the action of water for long periods. Residual kaolins[89] are of low plasticity, but water-transported kaolins of exactly the same chemical composition are quite plastic.

VI. Conclusion

Plasticity in clays, then, is due to the gelatinous state of matter, a state common to them because of their mode of origin. This gelatinous matter may be silicic acid gel, alumina gel, iron oxide gels, silicate gels, or organic gels. Two or more of these are usually present, and their effect will be further modified by adsorbed salts and the relative proportions of large and small grains, and to a limited extent by the shape of the grains.

The particular kind and amount of gelatinous matter present, the size and shape of grain, and the relative proportions of large and small grains, are important factors in determining the other related physical properties of tensile strength and air shrinkage.

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[89] Kaolin washed for the market.

White-Burning Clays of the Southern Appalachian States

BY JOEL H. WATKINS, WASHINGTON, D. C.

(New York Meeting, February, 1915)

THE terms kaolin, china clay, ball clay, and paper clay are more or less loosely and interchangeably applied to a large class of white-burning clays. These clays are made up chiefly of hydrous amorphous (colloidal) aluminum silicates with variable amounts of free silica and other impurities in small quantities. Occasionally also the term kaolinite is wrongly applied to a white-burning clay, for kaolinite is a mineral of definite chemical composition and definite crystalline form which occurs but sparingly in nature.

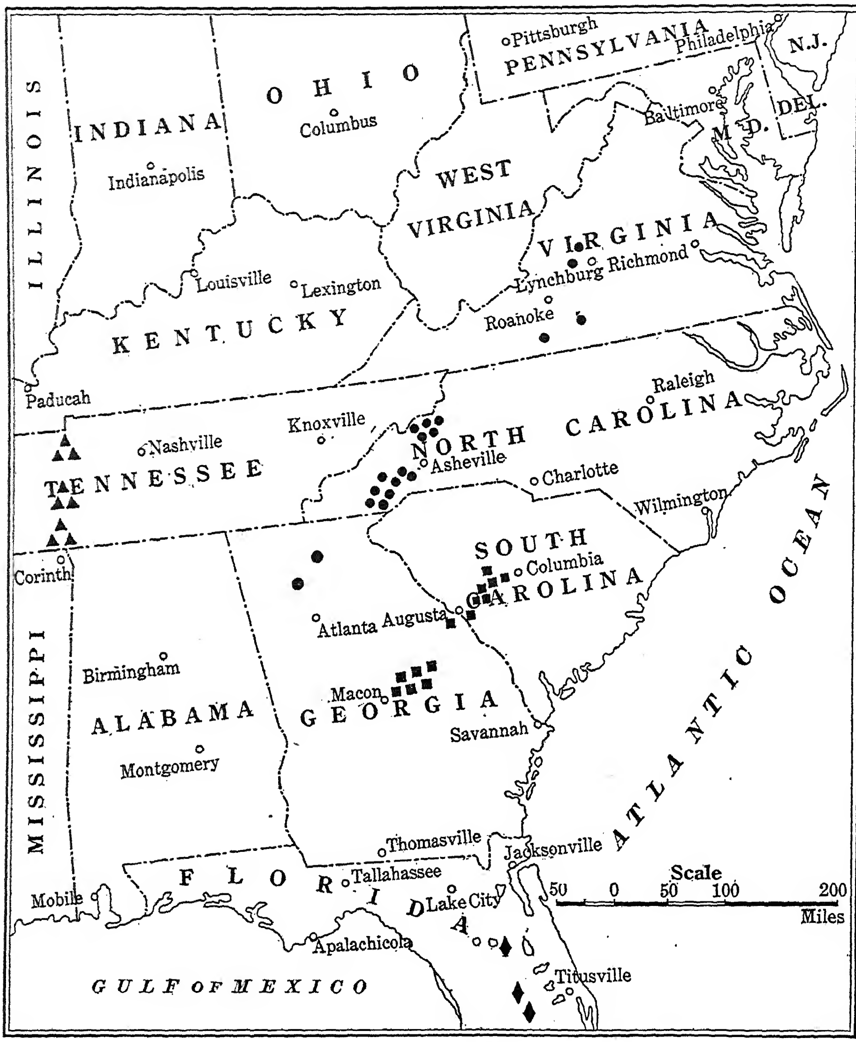
It is not the purpose of this paper, however, to define or classify clays, or to enter into a discussion of the chemistry of clays. It is rather a general description of the occurrence and methods of mining of a kindred group of clays, and a discussion of their future economic aspect.

In the Southern Appalachian States, the mining of white-burning clays has in recent years become an important industry. Although these clays are of several distinct types, which occur in as many localities under different geologic conditions, they are for the most part primarily of the same origin. They are essentially the same in elementary constituents, and are largely consumed in the same industrial arts. From the standpoint of a mining engineer, therefore, and for convenience in comparison and correlation, it should be interesting to discuss them under one heading.

KINDS OF CLAY

The most important types of clay which are now being actively mined in the Southern Appalachian States and with which this paper is particularly concerned are the North Carolina, Virginia, and Georgia kaolins or china clays (residual); the South Carolina and Georgia kaolins or paper clays (sedimentary); and the Florida and Tennessee plastic kaolins or ball clays (sedimentary). Other varieties which are interesting but which are as yet of little importance are the pocket deposits of white clay in Augusta County, Virginia, the bauxitic kaolins of Tennessee,

Georgia, and Alabama, and the halloysite deposits of Tennessee and Georgia. (Fig. 1.)



● RESIDUAL KAOLIN MINES, ■ SEDIMENTARY KAOLIN MINES,

♦ FLORIDA BALL CLAY MINES, ▲ TENNESSEE BALL CLAY MINES.

Fig. 1.—Map showing Geographic Distribution of Clay Mines in the Southern Appalachian States.

I. Residual Kaolins or China Clays

Origin.—The term kaolin is probably most correctly used when applied to white-burning residual clays which are derived directly from the

decomposition of feldspars in place, and which, when pure, conform closely to the chemical composition of the mineral kaolinite. The process of decomposition of feldspars, resulting in the formation of kaolin, is termed kaolinization. When kaolinization takes place to sufficient depth in highly feldspathic bodies of good dimensions, it often results in the formation of valuable deposits of china or paper clay. True pegmatite dikes probably afford the best examples of deposits of this character.

Distribution.—Pegmatite dikes occur abundantly throughout the crystalline area of the Southern Appalachian States. Workable deposits of kaolin are developed in Virginia, North Carolina, and Georgia.

North Carolina is at present by far the most important of the States which have produced residual kaolin. Pegmatite bodies of large and small size are so numerous over portions of Yancey, Mitchell, Jackson, Swain, and Macon counties that one might say the country rock has simply been saturated with granite juice. Mining is most active in these counties.

In Virginia, kaolin has been produced from pegmatites at only two localities: namely, about 3½ miles southeast of Henry, in Henry County, and about 8 miles northwest of Arrington, in Nelson County. Neither of these plants is being operated at this time. Other deposits in Virginia which have been developed to some extent are near Motley, in Pittsylvania County and near Forest Depot, in Bedford County.

In Georgia, kaolinized pegmatities have been prospected in Paulding, Pickens, Raburn, White and other counties of the Piedmont region, but, so far as known, there has been no production of residual kaolin in the State.

Geologic Relations.—In North Carolina, the pegmatite dikes that have proved to be of chief commercial importance are principally confined to the rough and mountainous region of the western part of the State. In their distribution they favor no particular topography, but are found alike on the slopes of high mountains and in the intervening valleys. They are intruded into granites, gneisses, and schists, but are more highly developed in the gneisses and schists, which offer less resistance along lines of rock cleavage. They may vary in width from a few inches up to several hundred feet, and in length from a few feet up to several miles. They are lenticular bodies which "swell" and "pinch" both laterally and vertically, the perpendicular dimension being more persistent. In the Appalachian States these dikes generally conform roughly to the foliation of the inclosing rocks, though they sometimes cut across the structure. In most cases the dikes are nearly vertical, though occasionally the dip is quite oblique. The inclosing rocks are pre-Cambrian in age, and of two distinct types, the Carolina gneiss and the Roan gneiss. The Carolina gneiss consists of a great series of mica-schist and garnet-

schist, mica-gneiss, garnet-gneiss, and cyanite-gneiss. The Roan gneiss consists mostly of hornblende-gneiss and hornblende-schist, generally forming long narrow bands and being less prominent than the Carolina gneiss. The pegmatites favor the Carolina gneiss, particularly near contacts with the Roan gneiss. Kaolinization in depth favors high ground which has gentle slopes and is least subjected to erosion. (Fig. 2.)

In Nelson and Amherst counties, Virginia, is a remarkable body of rock which has formerly been referred to as pegmatite, but has recently been shown by Watson and Taber[1] to be a syenite. This rock underlies an area of approximately 20 square miles, being 13 miles in length and

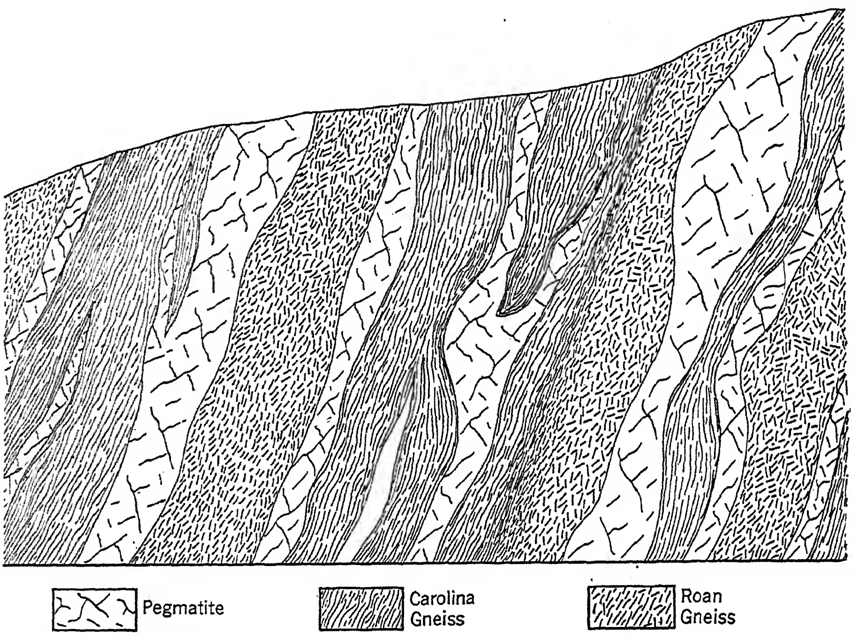


FIG. 2.—SECTION SHOWING IRREGULAR LENSE-SHAPED PEGMATITE DIKES AND RELATION TO INCLOSING ROCKS.

2½ miles in greatest width, with a general northeast-southwest trend. The central and much the largest portion of this intrusive body is made up almost entirely of feldspar with little quartz and essentially no ferro-magnesian silicates. In places the feldspathic facies of this mass is highly kaolinized, but the kaolin is not plastic, as there seems to be a considerable amount of admixed finely divided secondary silica. The kaolin is very white, however, and on account of the small amount of coarsely crystalline quartz and other minerals the percentage recovery of the material mined should be even higher than that of most North Carolina deposits. On account of its distance from the railroad, which is from

[1] Watson, Thomas L., and Taber, S.: Geology of the Titanium and Rutile and Apatite Deposits of Virginia, *Bulletin No.* 111—*A, Virginia Geological Survey* (1913).

8 to 10 miles, this field has been but little exploited for kaolin. These are probably among the most extensive deposits of commercial residual kaolin in America and with better transportation facilities should become of great economic importance.

Prospecting and Developing.—The districts in which commercial deposits of kaolin are known to occur have been little prospected except within a few miles of the railroads. Highly kaolinized pegmatites are rarely exposed at the surface in such a way that they can be readily recognized. They are invariably covered with several feet of dark-colored impure clay, stained with oxides of iron and decomposed vegetable matter. Loose fragments of white quartz and finely divided particles of mica in the soil are good guides for the prospector. Occasionally the white kaolin itself is exposed in hillside gullies. Little is known as to the extent of these deposits, but any one who visits the field cannot but be impressed with the extent of the areas and the number of pegmatites to be seen along rock exposures on stream courses.

Four important factors must be considered in exploiting these deposits as kaolin producers on a paying basis; namely, width and dip of dike, depth of kaolinization, percentage of recoverable kaolin, and distance from transportation. As stated above, 10 ft. is considered a minimum workable width, in which case it is desirable that the dike be as nearly vertical as possible. Depth of kaolinization is the next factor governing an estimate of the outside dimensions of a deposit. If these two factors are found to be in good proportion, the length of deposit can generally be counted on as sufficient to justify development. The outside dimensions of a deposit being approximated, the percentage of recoverable kaolin, determined by careful sampling and washing, furnishes the final factor for an estimate of kaolin in the ground. It is hardly possible for an entire deposit to average more than 40 per cent. kaolin, although parts of any deposit may have a segregation of kaolin running as high as 90 per cent. In one instance, where the deposit is more than 150 ft. wide, it is reported that only 15 per cent. of the material washed is recoverable as kaolin slip. A 25 per cent. recovery is considered a good average. No deposit has yet been worked which is at a greater distance than 4 miles from a railroad. In one instance a plant is being successfully operated where the refined kaolin has to be hauled in wagons a distance of 4 miles. Such a long haul, however, and particularly over mountain roads, greatly reduces the margin of profit for the producer. Tram roads have been used to some extent between washing plants and the railroads. Flumes are also used to advantage where surface conditions are favorable and the distance is not too great.

New deposits may be discovered at any time on ground which has been traversed repeatedly. As an example of this kind, one of the smaller plants in North Carolina, which was promoted on the prospects of deriving its kaolin from a dike about 8 ft. wide which had a dip of

about 45°, after producing 50 tons of kaolin, was forced to suspend operations. This company would have been a monumental failure had not one of its employees about that time discovered, 1½ miles from the plant, a kaolinized pegmatite about 200 ft. wide on ground which had been traversed repeatedly by other prospectors. The writer had the privilege of making a report on this property for the owners, and from 18 samples carefully taken from a number of shafts (Fig. 3) and test pits obtained the following percentage recovery of kaolin after washing. Samples Nos. 1 to 6 were taken at various depths in a 62-ft. shaft and Nos. 7 to 18 from test pits.

Sample No.	Sampling Depth, feet	Kaolin, Per Cent.
1	15	28.81
2	25	21.95
3	35	44.82
4	45	19.15
5	55	15.79
6	62	10.52
7	10	24.44
8	15	21.43
9	15	28.90
10	6	30.19
11	10	21.74
12	15	30.95
13	15	28.00
14	15	44.60
15	15	46.66
16	10	40.00
17	25	22.38
18	10	10.35

The average percentage recovery of kaolin from the 18 samples washed was 27.26, which is somewhat higher than should be expected from the actual mining and washing of the entire kaolinized portion. This deposit will probably prove to be one of the most extensive yet developed in North Carolina.

Mining and Preparation.—The mining and treatment of residual kaolin in the Appalachian Mountain region has been so thoroughly discussed by Watts[2] that only a summary of the essential details will be outlined in this paper. Emphasis will also be given some of the points advocated by Mr. Watts in regard to the future development of these deposits.

Where open-cut mining is possible, it is, of course, always the easiest, and has first preference. The method of mining kaolin by open circular shafts, however, which is exclusively practiced in the case of residual deposits, is both unique and economic (Fig. 4). Since in the zone of weathering the wall rock is decayed to about the same depth as the peg-

[2] Watts, A. S.: Mining and Treatment of Feldspar and Kaolin in the Southern Appalachian Region, *Bulletin No. 53, U. S. Bureau of Mines* (1913).

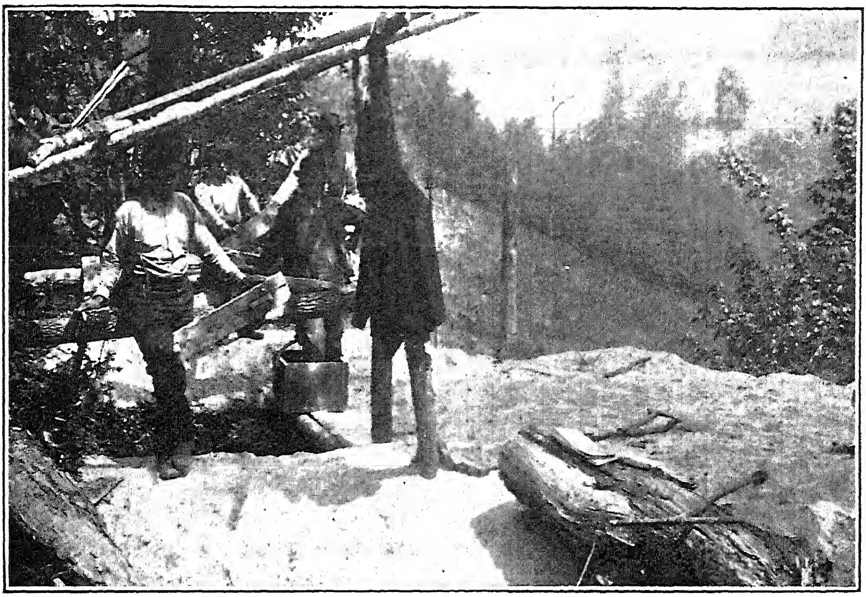


FIG. 3.—TYPICAL PROSPECT SHAFT, WESTERN NORTH CAROLINA RESIDUAL KAOLIN.

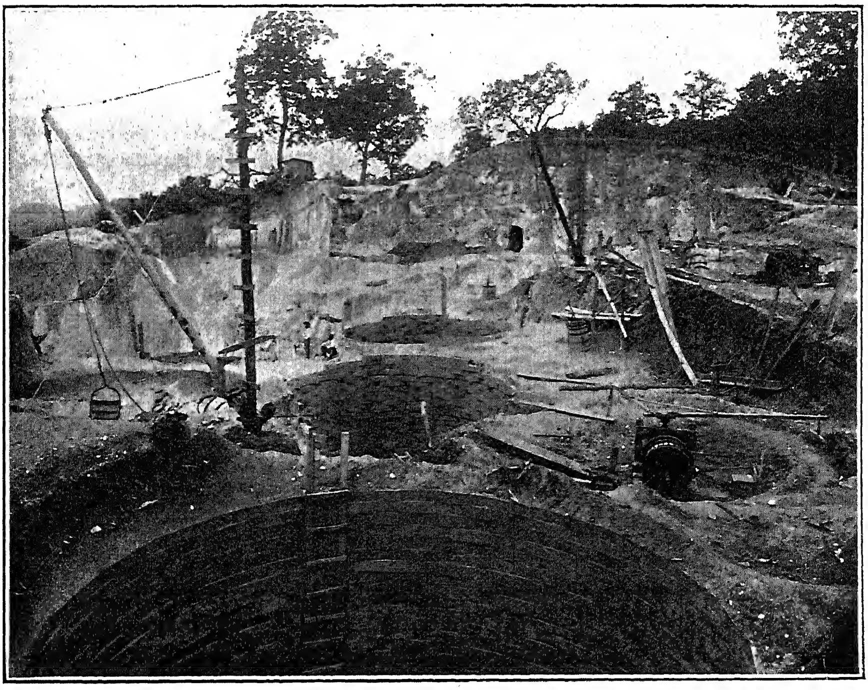


FIG. 4.—TYPICAL NORTH CAROLINA RESIDUAL KAOLIN MINE.

matites, underground mining by the usual methods is both difficult and dangerous. Open-cut mining is always carried forward until slides from the walls become troublesome. The open cuts vary in depth with width of dike from less than 10 ft., in very narrow ones, up to 40 ft. in those 100 ft. wide and more. The smaller dikes are almost invariably richer in feldspar and consequently when weathered are richer in kaolin. Kaolinization in some instances reaches a depth of more than 100 ft., but the average depth of workable deposits is probably nearer 50 ft. Most dikes have a dip of more than 70°, but when the dip is less than 70° and the width is less than 20 ft. the problem of winning the kaolin becomes complicated. Ten feet is probably a minimum workable width for a pegmatite high in kaolin which has a nearly vertical dip, and a dike of such dimensions can be worked to a depth of 50 ft. with a shaft 13 ft. in diameter. If a dike 10 ft. wide, however, has a dip of less than 60°, the foot wall would soon be penetrated by a vertical shaft.

It has not been found practicable to undertake mining by open circular shafts on an incline. In the wider dikes, shafts are started where open-cut work is left off. When a shaft reaches the depth at which the percentage of recoverable kaolin is too low to be of value, another shaft is started a short distance from the first. The space between the shafts is generally robbed through holes made by removing timbers at different levels. The abandoned shafts are filled with waste from other shafts and cuts, while stoping between the shafts and removal of timbers advances as the shaft is filled. In this way, all of the kaolin is won and the same timbers can be used repeatedly. The shafts vary in diameter from 13 to 20 ft., and several men can work in the bottom of each shaft without being crowded. Ordinary derricks with mast and boom operated by drum hoists are in common use. Half-barrel buckets, which hold about 500 lb. of material, are lowered into the open shaft and filled while other buckets are being hoisted and dumped in flumes or on waste heaps.

With the exception of quartz veins and occasional masses of semi-kaolinized feldspar within the pegmatites, all of the material mined can be won with pick and shovel. Massive quartz veins often several feet thick generally form one of the walls of the pegmatites, though sometimes found within the dike itself. They are difficult to move, and as there is little or no market for quartz in this territory it is for the present thrown in with other waste materials. Sheet mica of marketable dimensions and such minerals as pitchblende, beryl, and gems are always saved and in some cases go a long way toward paying the running expenses of the mine. Accessory minerals are uncertain factors, however, and should never be counted in the valuation of a kaolin mine.

As the mines are always on high ground the kaolinized material can generally be flumed to the washing or refining plant. These plants require a great deal of clear water and are generally located along streams and as near a railroad as possible. With sufficient fall and well-con-

structed flumes, it is possible to transport the material several miles to a washer.

The washing plants consist of a series of disintegrators, sand wheels, sand troughs, mica troughs, screens, settling tanks, agitators, and filter presses, which represent the different steps in separating kaolin slip from waste materials. After the refined kaolin is taken from the filter presses, it is dried either in open sheds, on steam tables, or in hot-air tunnels.

Analyses of North Carolina Residual Kaolins

(Analysis No. 1 by Ries, Nos. 2, 3, and 4 by Watts)

	1	2	3	4
SiO_2	45.40	46.95	44.00	45.20
Al_2O_3	39.34	37.24	40.79	38.45
Fe_2O_3	1.92	0.40	0.11	0.45
TiO_2	....	0.05	Trace	Trace
CaO	0.44	Trace	Trace	Trace
MgO	0.20	Trace	Trace	Trace
K_2O and Na_2O	0.52	0.73	0 62	0.65
H_2O	13.96	14.10	14.72	14.80
Total	101.78	99.47	100.24	99.55

II. Sedimentary Kaolins or Paper Clays

Distribution.—In the Cretaceous horizon of South Carolina and Georgia are extensive beds of white clay which have been actively mined at several points for a number of years. These Cretaceous beds are exposed in a narrow belt extending in a general northeast-southwest direction across the States. The principal counties in which important beds of kaolin have thus far been developed are Bibb, Twiggs, Wilkinson, Washington, Glascock, Jefferson, and Richmond counties, Georgia; and Aiken, Edgefield, Lexington, Richmond, and Kershaw counties, South Carolina. Mining is most active in the vicinity of Dry Branch, McIntyre, and Hephzibah, Georgia, and Aiken, Langley, and Bath, South Carolina. In all there are about 12 companies producing kaolin in the two States at this time.

Origin.—From a genetic standpoint these clays have been correctly termed sedimentary kaolins, since they represent an accumulation of finely divided particles of kaolin which have been deposited from suspension in water. The primary origin of these particles of kaolin is attributed to the residual clays in the crystalline area of the Southern Appalachian region. A large portion of this area is underlain with highly feldspathic rocks, such as granites, syenites, pegmatites, aplites, or some of the finer-grained acid intrusives. The residual clays derived from these rocks contain more or less kaolinized feldspar, which for a long period of years has been gradually taken up by surface erosion and

transported by streams to the sea. At some period during Cretaceous times, all of the conditions favorable to the concentration and deposition of these kaolin particles must have existed.

From a comparison of the analyses of these clays with analyses of the North Carolina residual kaolins, it will be seen that they are closely similar, the greatest variation being that the ferric oxide and titanium oxide are somewhat higher in sedimentary clays than in the residual

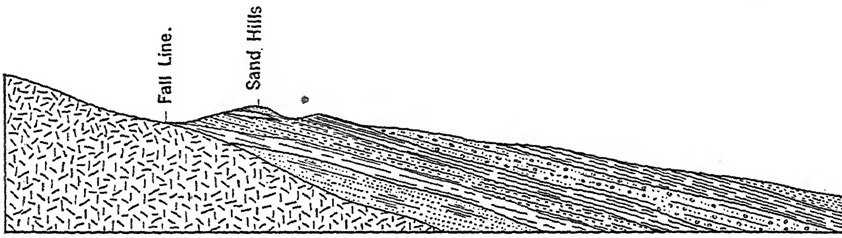


FIG. 5.—SECTION SHOWING RELATION BETWEEN COASTAL PLAIN SEDIMENTS AND UNDERLYING CRYSTALLINE ROCKS.

clays. This similarity in chemical composition, together with the presence of coarse white quartz sand and small particles of white mica, is additional evidence that the bedded kaolins are accumulations of transported materials derived from the residual of feldspathic rocks.

Geologic Relations.—The lower Cretaceous beds in South Carolina and Georgia comprise a series of cross-bedded sands, clays, and gravels which rest directly upon the crystalline rocks of the Piedmont Plateau (Figs. 5 and 6). The irregular surface contact between the Cretaceous

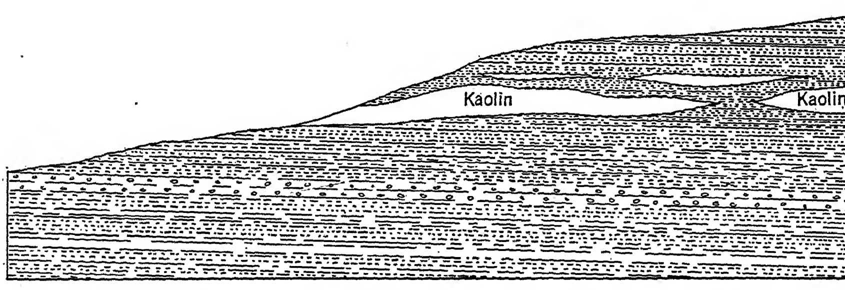


FIG. 6.—SECTION SHOWING GENERAL OUTLINE OF KAOLIN BEDS AND RELATION TO UNDERLYING AND OVERLYING SEDIMENTS.

and the crystallines is commonly known as the "fall line." This line is marked by a series of sand hills through which streams passing from the Piedmont Plateau to the Coastal Plain have cut small valleys, and the streams themselves often have considerable fall. The general elevation of the upland country of this narrow belt is between 400 and 600 ft. above sea level. In this division of the Cretaceous lying above the surface contact with the crystallines occur the great bedded deposits of white clay which are now so important to our domestic paper mills and pottery

plants. These beds of clay are by no means continuous, but may vary in thickness from a few inches up to 35 ft. within a comparatively short distance. In one instance the writer has observed in a railroad cut as many as three distinct beds of very white clay, one above the other, which were not more than 4 in. at greatest thickness, and which in a short distance thinned out to the thickness of a knife blade. The contact of the clay with the underlying and overlying sands is generally quite distinct, but in places the sand seems to grade into the clay for a few inches. The plane marking the contact with the underlying beds is much more regular than that marking the contact with overlying beds. Within the thickness of a single bed the clays may vary greatly in color, though the change in color is generally abrupt in different parts of the bed. For example, in one place the writer observed the following changes in color from top to bottom of a bed: Deep purple, 9 in.; light pink, 12 in.; light yellow, 2 ft.; white, 10 ft.; blue-gray, 3 ft. The overburden, which may be from zero to 50 ft. in thickness, consists of a series of reddish cross-bedded sands, sandy clays, and gravel with occasional thin ledges of hard sandy brown iron oxide.

Prospecting and Developing.—Probably the larger portion of these extensive clay deposits has already been eroded away and transported by surface waters. There still remain, however, wide areas of this sand-hill country which may be prospected for kaolin beds. As is usually the case, only those deposits which have easy access to railroad transportation have been developed. The outcrops are generally exposed on the slopes of hills, as the sandy beds are very loose and erosion is rapid. The surface exposures, however, only show the presence of the material and do not afford a fair idea of the dimensions of the deposits. Auger drills can be easily driven through the soft sand-clay sediments, and the actual thickness and character of a deposit can be satisfactorily determined by this means in a surprisingly short time.

Systematic prospecting in this field, with the aid of an engineer's level and an auger drill, should develop other workable deposits of kaolin at a relatively small expenditure.

Mining and Preparation.—The mining of sedimentary kaolins to the casual observer would seem but a simple operation; moving a few feet of soft, sandy overburden, then digging the snowy, white clay from the banks with pick and shovel, and laying it on shelves to dry. If the deposit always had access to transportation, if the overburden were always but a few feet, and if there were no rainfall, the operation might be a simple one indeed. The mine manager in this field, however, has his troubles, as well as those who have more complicated problems.

The kaolin is always first mined along the outcrop or where the overburden is least expensive to move. After the kaolin which is most easily won is exhausted, greater difficulties confront the manager in pushing his excavations further into the hill and moving the heavier

overburden. Open-cut mining is universally practiced, as underground methods have been unsuccessful. The overlying materials are so soft that the timbering would have to be water-tight in order to keep out the sand and iron-stained silt in rainy weather. The writer on one of his visits to the field witnessed an apparently successful attempt at underground mining with close timbering. A short time afterward he was advised by the local manager that two of the miners almost lost their lives by a quicksand cave-in, and the tunnel had to be abandoned.

The majority of the plants have direct railroad connections, but some of them use narrow-gauge railroads for about 2 miles.

Three methods of stripping are employed at different mines. The

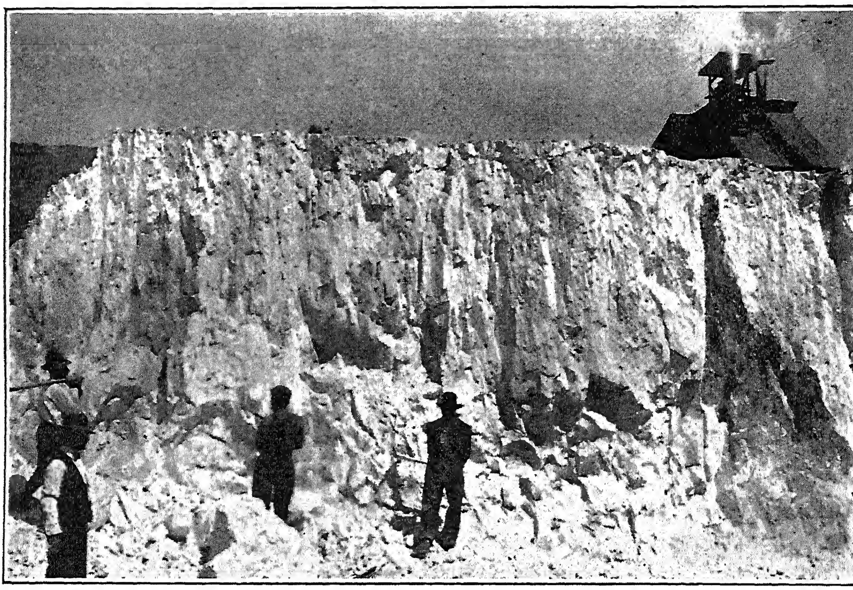


FIG. 7.—SOUTH CAROLINA SEDIMENTARY KAOLIN MINE, SHOWING 15 FT. OF FIRST-GRADE CLAY AFTER STRIPPING 30 FT. OF OVERBURDEN.

old way, which is still practiced at most of the mines, is with pick and shovel and mule carts. Several have steam shovels, and one is equipped with a cable drag line and bucket. Advantages are claimed for each of these methods, but it would be hard to say which is the most economical. In most cases the material has to be moved a considerable distance, so as not to interfere with future excavations. Rainy weather is never welcomed by the clay miner, as he not only has to suspend operations, but the rain washes iron-stained sand and silt over the face of his white clay bank. It also fills the pits with water, which has to be drained or pumped, and puts everything in such bad condition that it takes some time to put the mine in normal condition again. Generally, several grades of clay are produced from the same mine, and great care must be taken to

see that the different grades are kept separate. Native negro labor is almost universally employed.

Only a few of the mines have washing plants, and these are necessary only where sand is present within the kaolin bed. The washed kaolin, however, is so thoroughly mixed that it makes a more uniform product and for that reason commands a higher price. One of the companies has installed a pulverizing plant through which the kaolin is passed after drying. This is claimed to mix the clay from different portions of the bed thoroughly, and gives a more uniform product to the consumer. Most of the beds being worked are so free from sand that the clay does not require washing, but is simply dried before shipping. As the climate

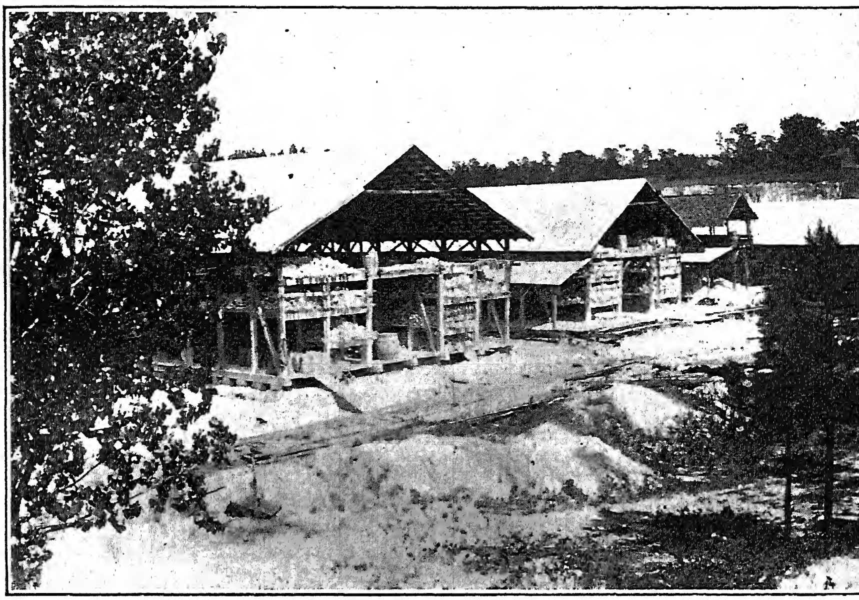


FIG. 8.—SOUTH CAROLINA SEDIMENTARY KAOLIN DRYING SHED, SHOWING KAOLIN READY FOR SHIPMENT IN HOGSHEADS AND IN BAGS.

is warm, open sheds with tiers of horizontal racks, upon which the lumps of clay are laid to dry, are in common use (Fig. 8). This, of course, is very economical and requires only one handling of the clay from the pits to the sheds and rehandling when shipping. Several sheds are maintained by most of the companies, as one can be filled while another is being emptied, and it is always well to keep a reserve supply of dried kaolin on hand for emergency orders. Steam driers are used by the companies that maintain washing plants, but this is probably because they have exhaust steam which can be used to advantage.

Kaolin is shipped loose in box cars, in bags, or in hogsheads, according to the desire of the consumer.

Analyses of South Carolina and Georgia Sedimentary Kaolins

(South Carolina Clays by Hardin and Georgia Clays by Everhart)

	South Carolina			Georgia		
	1	2	3	4	5	6
SiO_2	45.02	44.23	44.51	44.76	44.67	44 99
Al_2O_3..........	38.98	38.92	38.12	38.41	38.76	38.59
Fe_2O_3..........	0.77	2.31	1.75	0.63	0.85	2.11
TiO_2...........	0.85	1.21	1.11	1.37	1.37	1.04
CaO...........	0.03	0.12	0.06	0 20	Trace	Trace
MgO..........	0.07	Trace	Trace	0 09	0.08	0 05
Na_2O..........	0.55	0.26	0.41	0.09	Trace	0.24
K_2O...........	0.26	0 30	0.32	0.35	Trace	0.11
H_2O..........	13.58	12.90	13.45	14.68	14.86	12.97
Total........	100.11	100 25	99.73	100.58	100.59	100.25

III. Plastic Kaolins or Ball Clays

Florida Ball Clays

Distribution.—The white-burning clays of Florida which have thus far proved to be of economic importance are commonly known as the Florida ball clays. Exposures of these clays are confined to a belt extending from Putnam County south through Marion, Lake, Orange, and Polk counties. Mines have been operated only in the vicinity of Edgar, Putnam County, and in the vicinity of Richmond and Yalaha, Lake County. Other deposits of interest are reported in the vicinity of McMeekin, Putman County; Bartow Junction, Polk County, and along Palatlakaha River in Lake County.

Geologic Relations.—The area underlain with plastic kaolins occupies a portion of the Central Peninsular section of the State. Geologically this area is of Tertiary age and belongs to the Apalachicola group of the Oligocene series. The Apalachicola group consists of a series of interbedded sands, clays, marls, fullers earth, and limestone. These sediments rest directly upon the Ocala limestone and in the Central Peninsular portion of the State dip gently toward the Atlantic Ocean.

Origin and Occurrence.—The kaolin-bearing beds are covered with an overburden of coarse quartz pebbles and reddish sands which reach a thickness of 10 ft. or more. The kaolin occurs in beds of coarse grayish-white sand which reaches in places a thickness of more than 30 ft. Cross-bedding is distinct throughout the entire thickness of the bed and particularly near its upper limits. According to Sellards,[2] "A well put down by the Edgar Plastic Kaolin Co. is reported to have passed through

[2] Sellards, E. H.: *Second Annual Report, Florida Geological Survey*, p. 243 (1909).

coarse superficial sand 10 ft.; kaolin-bearing sand 30 or more feet; sticky blue clay with fullers earth beneath about 40 ft.; scarcely indurated shell stratum 20 ft.; the well terminated on a hard limestone rock at the depth of 90 ft."

Small particles of mica which are visible to the unaided eye are sparsely disseminated through the kaolin. This is evidence that the kaolin primarily had its origin in the residual clays of granitic rocks. It is very probable, however, that this kaolin was first deposited in Cretaceous times, and later eroded and transported to its present position. As this portion of Florida is several hundred miles farther from the crystalline area than is the Cretaceous horizon, it is reasonable to suppose

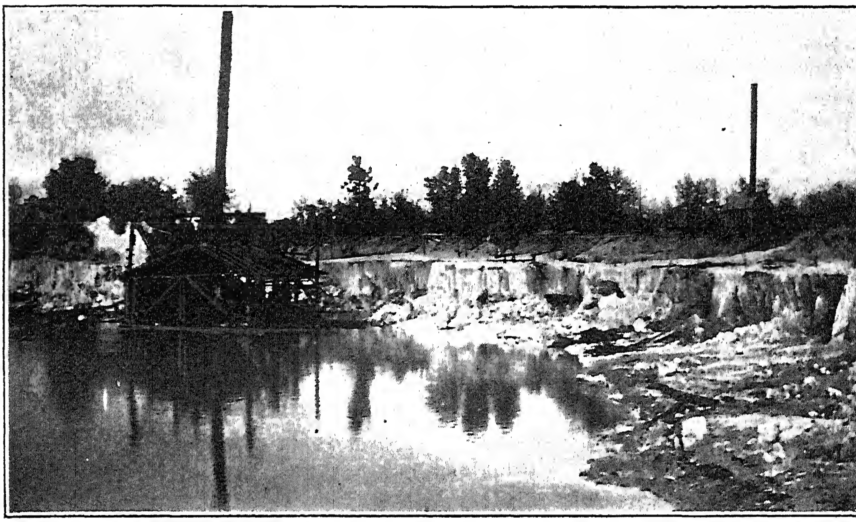


FIG. 9.—FLORIDA BALL CLAY MINE, SHOWING REMOVAL OF KAOLIN-BEARING SANDS WITH FLOATING DREDGE.

that the particles of kaolin held in suspension for so great a distance would be more finely divided than those which were deposited in Cretaceous beds. This to some degree may account for the fact that the Florida clays are more plastic than the Cretaceous clays.

Mining and Preparation.—Unlike the North Carolina residual clays and the Georgia sedimentary clays, the Florida clays are mined by dredging (Fig. 9). As the country is very flat, and any open-cut work necessarily makes a depression in which water soon accumulates, dredging is naturally the most economic method of winning the clay. However, it is probably due to the fact that the beds are about 70 per cent. sand that dredging is successful. If these clays were as compact and stiff as the Georgia clays, they would not soak up well or yield to a suction pump. The recoverable kaolin rarely represents more than 30 per cent. of the material washed, and is reported to average not more than 20 per

cent. After the overburden is removed the kaolin-bearing sand is liftec by the dredge and forced to a sufficient elevation above the washing plant From this point it flows by gravity through a series of settling tanks by which the sand and particles of mica are separated from the clay From the settling tanks the kaolin slip passes to the filter presses and tc the drying shed in the usual way.

Analyses of Florida Ball Clays

(No. 1, washed clay from Edgar; C. Langenbeck, analyst. No. 2, washed clay from Palatlakaha River; H. Ries, analyst.)

	1	2
Silica (SiO_2)	45.39	46.11
Alumina (Al_2O_3)	39.19	39.50
Ferric oxide (Fe_2O_3)	0.45	0.35
Lime (CaO)	0.51	....
Magnesia (MgO)	0.29	0.13
Alkalies (Na_2O,K_2O)	0.83	....
Water (H_2O)	14.01	13.78
Sulphur trioxide (SO_3)	....	0.07
Total	100.67	99.94

Tennessee Ball Clays

Distribution.—The mining of Tennessee ball clay is confined to a narrow strip of land extending across the western portion of the State in a general direction which is about 20° east of north. This belt, beginning at the north, passes through Henry, Carroll, Henderson, Madison, Chester, and Hardeman counties. Henry, Madison, and Carroll counties have been the most important producers of ball clay up to this time. Mining for shipment to potteries has been most active in the vicinity of Whitlock, Paris, and Henry, Henry County; McKenzie, Hico, and Hollow Rock, Carroll County, and near Pinson in Madison County.

Geologic Relations and Origin.—Probably the only clays in western Tennessee which might be termed white-burning clays are those plastic clays which are consumed by some of the white-ware pottery plants in their general mix. The best grades of this variety of clay are reported to burn to a cream white. As these clays occur associated with several other varieties of clay in the same geologic horizon and often in the same lense-shaped deposits, it is difficult to differentiate and define them. The other varieties, which can hardly be classed as white-burning, are stoneware clay, sagger clay, fire clay, and wad.

The most important clays belong to the LaGrange series of the Tertiary, although some occur in the Ripley sands of the Cretaceous. In geologic position they are very nearly the same as the Cretaceous clays of South Carolina and Georgia, only the Cretaceous beds in this

area rest upon Paleozoic strata instead of upon the older crystalline rocks. The workable clay deposits of Tennessee generally cover but a few acres and are not nearly so continuous and uniform in character as those of South Carolina and Georgia. They are overlain with reddish sands and sand-clay beds, which vary in thickness from a few inches up to 30 ft. and more, according to the amount of erosion.

Though the Tennessee ball clays are somewhat different in chemical composition and physical properties from the South Carolina and Georgia paper clays, it is possible that they may to some degree have the same origin. Nelson[3] attributes the primary origin of the clay substance embodied in these deposits to the disintegration of chert and the weathering of limestone in nearby Paleozoic rocks, but does not account for the presence of particles of white mica disseminated through the clay. Though this area seems far removed from the crystalline area of the Southern Appalachian region, it should be remembered that the waters of the Tennessee River, which must have once deposited their burden along this coast line, have their origin far into the crystalline area. Besides this, beds of white clay similar to those described above occur almost continuously in the Cretaceous from Tennessee through Mississippi and Alabama to Georgia. It would seem that a part of the clay may have had its origin in the residual clays of the crystalline rocks.

Mining and Preparation.—As the occurrence is very similar, practically the same methods of mining clay are practiced in Tennessee as in Georgia and South Carolina. The selling price, however, is not as high as that of the Georgia clays and therefore does not warrant washing or drying. It is shipped in box cars direct from the mines to the consumer.

Analyses of Three Different Grades of Tennessee Ball Clay

(Carl N. Zieme, analyst)

	1	2	3
Volatile	14.16	13.94	10.92
Silica	48.72	47.26	53.57
Alumina	32.89	35.85	32.82
Iron oxide	2.25	1.01	1.31
Calcium	0.60	0.58	0.62
Magnesium	0.65	0.68	0.38
Potassium	0.98	0.74	0.59
Sodium	0.14	0.45	0.21
Total	100.39	100.51	100.42

[3] Nelson, Wilbur A.: Clay Deposits of West Tennessee, *Bulletin No. 5, Tennessee State Geological Survey.*

IV. Miscellaneous Clays

Bauxitic Clay.—Associated with the bauxite deposits of Georgia, Alabama, and Tennessee are irregular deposits of clay, locally known as bauxitic clay. In many instances this material is very white and approaches kaolin in chemical composition. The clay occurs as horses in, or as irregular circular masses surrounding the deposits of bauxite. The contact with the bauxite may be sharp or the clay may grade into the bauxite. In the latter case the clay is always high in alumina. So far as is known, no serious attempt has ever been made to develop these clays commercially, but on account of their extent and physical properties they may yet prove to be of economic value.

Halloysite.—Halloysite is a clay-like substance which conforms closely to the chemical composition of kaolin, but has a considerably higher percentage of combined water. It is often waxy in appearance and usually has a greenish or buff color, sometimes white. Besides being associated with the bauxitic clays in small quantities, halloysite also occurs in pockety lenticular beds underlying the Fort Payne chert in Chatooga and Walker counties, Georgia. Other beds quite similar to those in Chatooga and Walker counties have recently been discovered about 8 miles west of Cleveland, in Bradley County, Tennessee. Samples of this halloysite have been burned repeatedly by the pottery manufacturers, and in every instance the color and vitrifying properties of the clay are reported to be equal to those of the highest-grade china clays. The uncertainty of the supply has probably restricted the demand for this material.

An interesting blue-black clay was recently found by the writer in the vicinity of Anniston, Ala. This clay is being tried out by several companies at present. A preliminary test by the Bureau of Standards at its clay-testing plant in Pittsburgh, Pa., is reported about as follows: The clay is highly plastic, it burns to a white color, which is equal to or superior to that of commercial ball clays, and inferior to that of commercial kaolins. The clay is not as highly vitrified at 1,200° C. as ball clay, but may have value as a substitute for a portion of the ball clay in the production of ceramic mixtures. To all appearances this clay is a Coastal Plain deposit, and the only shaft which has been put down passes through 30 ft. of the material. It is reported by some of the potteries that have made preliminary tests that it resembles the German ball clays in appearance and physical properties.

Uses

Few mineral substances have a wider range of commercial application than white clay. By far the largest portion of the white clay mined is consumed in the manufacture of pottery or white-burned products,

which includes white-ware and cream-colored ware; bone, delft, and belleck china ware; sanitary ware, porcelain electrical supplies, architectural terra cotta, floor and wall tiles, and miscellaneous articles. The paper mills are probably the next largest consumers of white clay. In addition to this a large amount of white clay is used as a filler in paints, plasters, pastes, putty and school crayons.

Production

The latest and most accurate figures on the production of white-burning clays in the United States are those published by the U. S. Geological Survey in the non-metallic section of *Mineral Resources* for 1913. Extracts from these figures are tabulated below:

Production of White-Burning Clays in the United States in 1913

States	Kaolin		Paper Clay		Ball Clay		Total		Average Value per Ton
	Tons	Value	Tons	Value	Tons	Value	Tons	Value	
North Carolina.	16,332	$139,629	....	....	....	....	16,332	$139,629	$8.55
South Carolina.	....	....	31,568	$120,520	....	....	31,568	120,520	3.82
Georgia......	....	....	69,740	299,110	....	....	69,740	299,110	4.28
Florida.......	....	....	....	....	[a]35,620	$143,650	[a]35,620	143,650	[a]4.03
Tennessee....	....	....	....	....	29,258	85,500	29,258	85,500	2.92
Total........	16,332	$139,629	101,308	$419,630	64,878	$229,150	182,518	$788,409	$4 32
Other States..	12,502	$95,828	....	....	2,256	$8,522	14,758	$104,350	$7.07

[a] Including some from other States.

It will be seen that North Carolina, South Carolina, Georgia, Florida, and Tennessee produced 182,518 tons, valued at $788,409, while other States produced only 14,758 tons, valued at $104,350. The highest average price per ton at the mines $8.55, was for North Carolina residual kaolins, while the lowest price $2.92, was for Tennessee ball clays. The average price per ton for all white clays produced in the five States referred to was $4.32.

Economic Aspect

In 1913 the United States produced white-burned products which had a total value of $31,443,450, and in the same year imported white-burned ware valued at $9,340,890. The total export of white ware from the United States in 1913 was valued at $149,281; thus showing that the total value of the white-burned pottery products entered for consumption in the United States in 1913 was $40,635,059.

In 1913, the imports of kaolin or china clay into the United States amounted to 268,666 short tons, valued at $1,623,993, with an average value of $6.04 per ton. According to Parsons,[4] "This kaolin displaced a like amount of domestic raw material, which, if properly handled, has no superior." Against this, in the same year, only 197,276 short tons of kaolin and ball clay were produced in this country, which had a total value of $893,259 and an average value of $4.53 per ton. Thus it may be seen that our domestic production of white-burning clays is only equal to about two-thirds of our imports in quantity, and half of our imports in value.

In consideration of the wonderful reserves of high-grade clays in the territory described above, it would seem that the United States, instead of importing annually large amounts of clay, should produce enough to supply her own demands, and even be a competitor in foreign markets. There are several good reasons, however, for this great difference between imports and domestic production, but only reasons which are due to existing conditions, of which can all be easily overcome.

One of the principal objections made by American potters to domestic pottery clays is that the product is not uniform in character, or in other words, has not been properly prepared and classified before shipping. The potter justly demands a uniform product under guarantee, since the slightest variation in his ceramic mixture may cause the entire plant to be shut down until the irregularity is discovered and eliminated. Such variations may be of either chemical or physical character and may result in a lack of sufficient plasticity, a change in color on burning, a change in temperature at point of vitrification, a change in shrinkage, a tendency to warp and crack, and a failure to take the glaze.

To meet the requirements of the potter, therefore, the U. S. Bureau of Mines has wisely advocated a central depot where the output of a number of mines may be properly mixed and graded under the supervision of a trained ceramic chemist. Such a plant, if successfully engineered, should practically eliminate the present practice of mining to-day and shipping to-morrow, and should greatly stimulate the mining of kaolin by smaller operators. The output should demand a higher price than the clay is now bringing, and should capture the trade of our domestic potteries.

Such a plant, or holding company, could be operated successfully not only in the North Carolina residual kaolin district, but also in the South Carolina and Georgia sedimentary kaolin districts.

Another reason why American white clays are not more extensively used is because the clay mines, for the most part, are so remote from the

[4] Parsons, Charles L.: Preface to Mining and Treatment of Feldspar and Kaolin in the Southern Appalachian Region, *Bulletin No.* 53, *U. S. Bureau of Mines* (1913).

manufacturing centers. Practically none of the white clays now being mined in the Southern Appalachian States are consumed in these States, and the bulk of the product has to be shipped by rail, more than 400 miles in the case of North Carolina kaolins, more than 600 miles in the case of South Carolina and Georgia kaolins, and more than 700 miles in the case of Florida ball clays. If these clays were properly prepared, and could be sold under guarantee to the pottery plants, there is no doubt that the potter would get closer to the source of his raw materials.

Estimates submitted by a number of clay miners in the North Carolina field show that this material, under favorable conditions, can be produced at a cost ranging from \$2 to \$4 per ton, and has a selling price of from \$8 to \$9 per ton at the mines. In the South Carolina and Georgia field estimates were submitted showing a range in cost of production of from \$1.50 to \$3 per ton with a selling price of from \$4 to \$6 per ton. The writer was not able to obtain figures on the cost of production of Florida ball clay, but, considering the great thickness of the clay-bearing beds and the economical methods of mining, it would be reasonable to assume that the margins of profit should compare favorably with those of the North Carolina and Georgia mines.

In view of the large amount of white clay consumed by American manufacturers, and the growing demand for domestic clays, it would seem that the clay-mining business in this country is almost in its infancy. Also, a review of the distribution of the clay resources in the United States would point to the fact that America must in the future look to the South for her supply of domestic white-burning clays.

Discussion

H. Ries, Ithaca, N. Y.—It is interesting to contemplate to what extent the American pottery industry would be affected in case the English supply of china clay were shut off for a considerable period of time. The English deposits are very large, and are worked on a most extensive scale. They are more easily and probably more cheaply worked than our American deposits, which are of vein-like nature. The American producer of washed clay is not always as careful as the foreign miner and washer of this material, although he has improved in recent years.

Our American washing plants are, furthermore, not as economical of material as it seems to me they might be, for the reason that more or less of the finest clay is often allowed to overflow, due to insufficient settling tank capacity. This fact is often shown by the milky character of the streams into which the waste water is discharged.

The Origin of the Louisiana and East Texas Salines

BY EDWARD G. NORTON, CHICAGO, ILL.

(New York Meeting, February, 1915)

THE salt deposits of the Mississippi Embayment region present a problem of origin so genetically related to the larger problem of the stratigraphy and structure of the region that a discussion of the one necessarily involves or assumes an understanding of the other. For this reason, I shall summarize briefly our knowledge of this region, emphasizing the facts related to the subject of this paper. In so doing I shall make liberal use of the reports of the National and State surveys.

Geographically that section of the United States known as the Mississippi Embayment region represents a great V-shaped area, extending from near Cairo, Ill., to the Gulf of Mexico, and including Louisiana, eastern Texas, all that part of Arkansas bordering the Ouachita and Ozark mountains, western Tennessee, Mississippi, and a part of Alabama.

This great section Professor Harris has described as structurally a pitching trough, its medial line being well represented by the Mississippi River.

Outcropping along the border of this area are the Cretaceous and Tertiary formations, unconformably laid down on the hard Paleozoic rock sheets that show more or less plication, deformation, and faulting, depending upon their proximity to the nearby major uplifts.

East and west of the Mississippi River the Cretaceous and Tertiary beds dip river- and coastward at angles greater than the slope of the land. "Waters entering along the outcropping edges of the Tertiaries and Cretaceous and following bedding planes southward are soon thousands of feet beneath the surface."

As the sedimentation of the region progressed, faulting and slips occurred along planes of pre-Cretaceous weakness. Whether these movements were genetically related to the loading of this V-shaped area, as Professor Harris seems to think, is a question still to be settled.

The accompanying map, Fig. 1, was taken from *Bulletin No.* 7 of the Louisiana Geological Survey. It shows the regional distribution of the salines, as well as an alignment of the domes, which has been assumed to indicate faulting.

That some of these fault zones exist, as shown on the map, is quite certain. Especially is this believed to be true of those running northeast and southwest. Others are to a great extent conjectural; and while some doubtless exist others have been assumed without much reason, and seem to impose upon nature a criss-cross arrangement of fault lines of such great regularity as to exceed her tolerance for geometrical design.

The arrangement of these domes in the vicinity of the "Sabine Peninsula" is noteworthy, as is the alignment of the Five Islands.

The stratigraphy and history of this region have been so well summarized by Veatch in *Professional Paper No.* 46 of the U. S. Geological Survey that I introduce his table here.

It will be noticed that in this statement the land period represented by the ligniferous sands and clays of the Bingen formation is separated from the ligniferous sands and the shales of the Tertiary by the marine beds of the upper Cretaceous. Penetrating the Cretaceous and Tertiary beds are cores of rock salt several hundred feet in diameter and from 2,000 to 3,000 ft. in thickness. These cores of salt are often capped by a highly crystalline limestone or marble, that is quite as remarkable and difficult of explanation in this alluvial country as is the occurrence of the salt itself. Around them are found deposits of gypsum. The further occurrence of sulphur, dolomite, sulphur dioxide, hydrogen sulphide, hot saline waters, large accumulations of oil and gas, are all peculiar to these localities.

The disturbance of the clays and shales in the immediate vicinity of these deposits indicates faulting; and the alignment of the salines would suggest that their distribution is along such lines of weakness.

If salt in regular bedding exists in Louisiana and Texas it has never been penetrated by the drill. If we assume that it exists thus at depths greater than have been reached, and has been elevated to the surface by an anticlinal development, the assumption is not supported by evidence that such mountain-building forces have been at work. The sediments throughout Louisiana and Texas show little disturbance.

At present there seems to be great unanimity of opinion among geologists that the salt, crystalline limestone, dolomite, and gypsum are secondary, and not indigenous to the formations in which they occur.

The origin of these salines has been the subject of much interesting speculation. An outline of the different hypotheses that have been advanced in explanation of these unique deposits will be found in *Bulletin No.* 7 of the Geological Survey of Louisiana.

In investigating them one is impressed by the fact that chemical reactions which appear to be taking place at the surface result in the formation of the same compounds that we find ranging from the surface down to depths of 1,500 or 2,000 ft. Moreover, proximity to the surface seems to favor such chemical reactions, if it does not actually make them possible.

Geologic subdivisions				Deposition: Thickness		Deposition: Character	
System	Series	Group	Formation	*Feet.*		Character	
Quaternary.	Recent.		Alluvium.	0– 20+		Veneer of sand, silt and clay on flood plains.	
				0– 25		Abnormal deposits of silt in Red River Valley resulting from the obstruction by the "great raft."	
	Pleistocene.					Formation of natural mounds.	
			Port Hudson formation.	0–200		Marine deposits on the coast and fluviatile deposits in the river valleys, partly filling the broad valleys developed in the preceding erosion cycle.	
						Rearrangement of surficial sands and gravels at new levels as erosion progressed.	
Tertiary.	Pliocene.		Lafayette formation.	[a]0– 50		A mantle of silt, sand, and gravel spread by combined marine and river action over the relatively even surface of the Coastal Plain and in the tributary valleys.	
	Miocene.						
	Oligocene.		Fleming clay.	±260		Green calcareous clays, with a few brackish-water fossils.	
			Catahoula formation.	1,000–1,200		Near-shore deposits; sandstones occasionally quartzitic, and green clays, with fresh-water shells and land plants.	
			Vicksburg formation.	100– 200		Limestones and calcareous, somewhat lignitiferous, clays, containing marine shells	
	Eocene.[b]	Undifferentiated Eocene.[b]	Jackson formation.	200– 550		Highly fossiliferous shallow-water marine sandy calcareous clay.	
			Cockfield member of Claiborne.[c]	400– 500		Lignitiferous sands and clays, with land plants.	Landward these formations merge into lignitiferous sands and clays without distinctive marine fossils.
			Claiborne formation.	200– 500		Fossiliferous sandy clay, containing shallow-water marine shells.	
			Sabine formation.	300– 900		Lignitiferous sands and clays, with plants and occasional beds of marine shells.	
			Midway formation.	20– 260		Limestones and black calcareous clays.	
Cretaceous.	Gulf series (upper Cretaceous).		Arkadelphia clay.	300– 600		Black laminated clays, with marine fossils in lower portions.	
			Nacatoch sand.	100– 160		Sand, with occasional calcareous quartzitic layers containing marine shells.	
			Marlbrook formation	150– 750		Very calcareous clay, with marine fossils.	
			Annona chalk.	+100		White chalk.	
			Brownstown formation.	150– 500		Blue calcareous clay, with marine fossils.	
		Bingen formation.	Sub-Clarksville sand.[d]	50–100	40–700+	Water-bearing sand	Lignitiferous sand and clays, with plant remains (Bingen formation).
			Eagle Ford clay.[d]	0-600		Blue calcareous clay.	
			Woodbine formation.	+500		Lignitiferous sands and clays, with plant remains.	
	Comanche series (lower Cretaceous)		Washita group.	0– 400		Dark calcareous clays, with clayey limestone beds.	
			Goodlaond imestne.	25– 30		Massive white limestone.	
			Trinity formation.	500– 600		Yellow, somewhat argillaceous pack sands, changing to the east to calcareous clays, with thin beds of limestone and gypsum.	
Pre-Cretaceous.			An immensely long period composed of the following principal stages in the order leveled these folds. On this planed-off surface the Cretaceous beds were deposited.				

a Normal thickness in northern Louisiana and southern Arkansas not known because of the widespread and irregular deposition. In southern Louisiana these beds are much thicker than here given.

b The Jackson, Claiborne, and Sabine formations, which are fossiliferous and distinct in central Louisiana, grade into lignitiferous beds containing no distinct fossils as they go northward. In the region under discussion the fossiliferous Jackson limits this lignitiferous complex above. Still farther north, however, the Jackson also grows lignitiferous and merges with the rest. The Midway, likewise, in the

From Professional Paper No. 46 U. S. Geological Survey.

Characteristic activities	
Degradation	Deformation
General degradation of the hill lands. Along Red River in Louisiana the resurrection of buried channels and the drainage of lakes produced by the "great raft" On the Sabine the partial wearing out of shoals produced by the recent movement of the Angelina-Caldwell flexure.	A slight upward movement at the west end of the Rockland-Vicksburg flexure is producing rapids on Sabine and Angelina rivers. A recent movement of 25 feet along the line of the Red River-Alabama Landing fault has resulted in the swamping of Ouachita River Valley to a point above the mouth of Bayou Moro in Arkansas
Partial removal of valley fillings and production of present flood plains and principal terraces.	
Long and complex period of erosion, with the land 100 feet higher than to-day, in which the formations of the Coastal Plain were profoundly dissected and the major features of the present topography produced.	After the main development of the Angelina-Caldwell flexure the beds were faulted along a line extending from a point near Denison, Tex, through Alabama Landing, Union Parish, La. The downthrow of this fault is to the north and the break approximately 600 feet.
A period of erosion, probably composed of several stages, in which the Coastal Plain in this region was essentially base-leveled.	The low fold which extends from the vicinity of Angelina County, Tex., to a point north of Vicksburg, Miss., and which is now a line of weakness, began to develop in late Oligocene or early Miocene time North of this line the older beds are now nearly horizontal; to the south they dip at a rate of from 35 to 150 feet per mile.
	The domes developed during late Cretaceous and early Eocene time show a slight movement in post-Claiborne time, but the amount is very small when compared with the initial movements.
Beds separated by a pronounced break in the fauna, which is, at present the only indication of a very serious break in sedimentation.	The great north-and-south fault of the Coastal Plain of Texas (the Balcones fault) developed late in the Cretaceous. In Louisiana peculiar domes or four-sided folds were produced and reached their major development in the late Cretaceous or early Eocene. About the same time masses of igneous rocks of limited area were intruded into the Paleozoic rocks and Coastal Plain beds in southern Arkansas. In central Texas similar occurrences took place as early as the Austin epoch. The Louisiana and northeastern Texas domes are thought to be due to the upthrust of similar igneous intrusions.
The Bingen and Woodbine sands indicate deposits very near shore and the fauna changes sharply at this point, but there is no evidence of a pronounced unconformity or a land period of great length.	

given: (1) Deposition, (2) profound folding and faulting, (3) long erosion, which essentially base-

upper embayment region shows a decidedly lignitiferous tendency and may in places merge with the lignitiferous time equivalents of the other Eocene beds.

c A group without distinctive marine fossils, probably almost wholly of Claiborne age.

d These beds do not outcrop in the Arkansas area, except as they may be represented as littoral deposits in the upper part of the Bingen sand, and therefore need not be considered in wells drilled near the northern edge of the Brownstown formation; but they will be found farther south and must there be allowed for in well calculations.

The seams of lignite common in well records down to depths of 1,800 ft. indicate that the country at that time was not entirely sub-

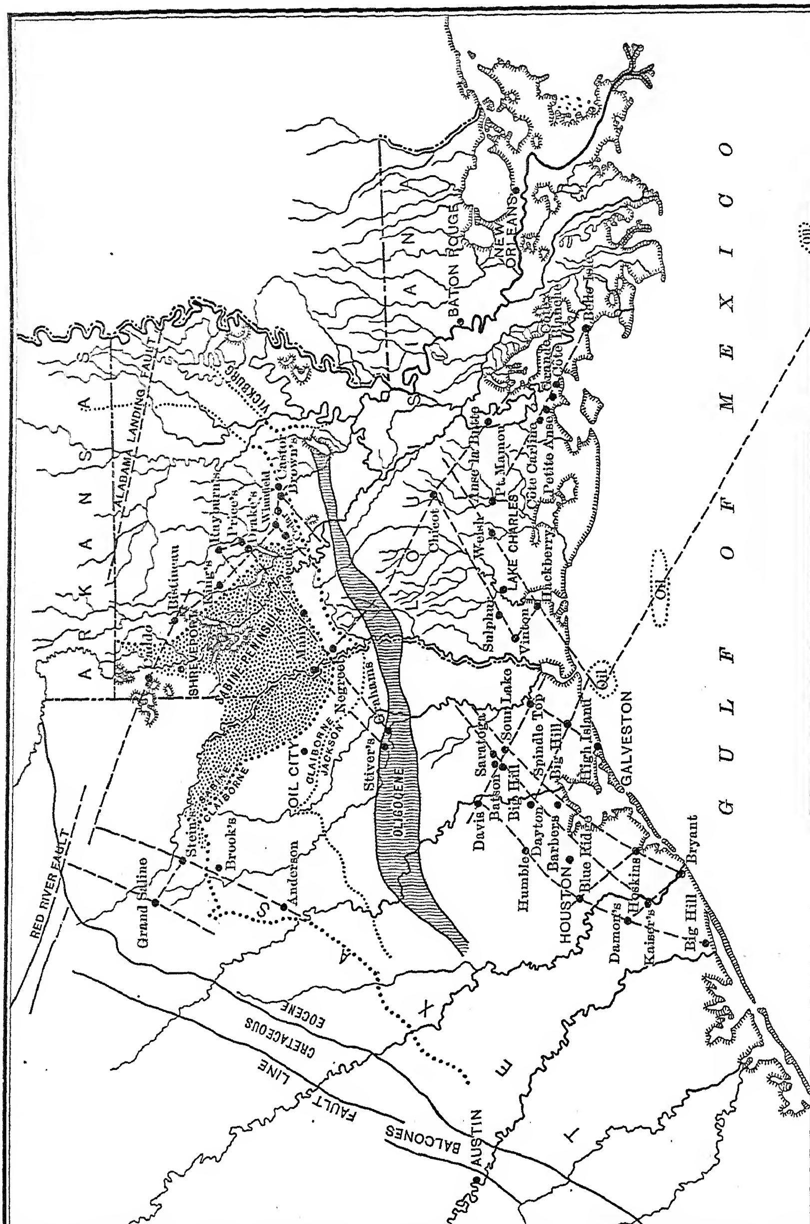


FIG. 1.—Map Showing Distribution of Salines and Alignment of Domes in Mississippi Embayment Region. *Bulletin No. 7, Louisiana Geological Survey*

merged and that many of the shale beds represent the flood-plain deposits of a Tertiary river system. Such a conclusion is further strength-

ened by the fact that, in northern Louisiana, cross-bedded Tertiary sands are by no means uncommon.

Since the cores of salt and gypsum farther south presumably penetrate these formations, the questions arise: Have not the cores been formed contemporaneously with the beds? Have not these salines for a very long time presented about the same appearance as now? Is the chemical action that is at present taking place very different from what it has been in the past? Are not these salines the product of the same forces as are now active?

Hayes and Kennedy, in describing High Island,[1] mention the fact that

> "At various points about the margin of the island are salt springs, which give rise to the trembling marshes. The water contains much calcium carbonate in solution, which is precipitated by evaporation about the spring vents and forms a crust over the surface of low mounds, being held together by the coarse marsh grasses."

A. C. Veatch[2] calls attention at Drakes saline to an "outcrop of grey, granular, sandy limestone containing very imperfect plant impressions."

Vaughan has made the following statement: "Near Atlanta in Winn parish, there outcrops a hard, blue limestone, which is traversed by minute fissures. In these fissures a small amount of gold is found." Veatch adds that "this must have been a near shore deposit, for it contains[3] the impressions of dicotyledonous leaves."

Limestone containing leaves I would not regard as of marine origin, or as having been formed in the ordinary way. Inasmuch as its occurrence is confined to these salines, the action that is at present taking place at High Island would seem best to explain its origin.

A well section at Rayburn's salt works[4] is interesting:

A. Whitish mud of the lick, with ferruginous spots and at its base frequently bearing balls of calcite.. 6 ft.
B. Siliceous gravel often cemented into a conglomerate by crystallized calcite .... 6 to 7 ft.
C. Greyish or white crystalline limestone, horizontally banded, fragile, often covered with 5 to 6 in. of crystallized aggregates of calcite, on a dark banded base of the same.. 6 ft.
D. Dense, banded gypsum, pure.................................... 2 ft.

The siliceous gravel that has been cemented into a conglomerate by crystallized calcite is supposed to be of Pliocene age.

It would appear from the literature, as well as from my own investigation, that calcium carbonate, limestone containing plant impressions, gypsum, sulphur, and probably salt, are being formed at or near the surface at the present time.

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[1] *Bulletin No.* 212, *U. S. Geological Survey*, p. 122 (1903).

[2] *Report of the Geological Survey of Louisiana for* 1902, p. 60.

[3] *Report of the Geological Survey of Louisiana for* 1899, p. 60.

[4] *Report of the Geological Survey of Louisiana for* 1902, p. 73.

Inasmuch as our ingenuity is about exhausted in an attempt to explain the occurrence and origin of this same group of minerals underground, the assumption would in a measure seem to be justified that they have for the most part been formed at or near the surface and that these cores of salt and gypsum have been built up contemporaneously with the sedimentation of the region.

The hypothesis that I have formulated in an attempt to harmonize the field evidence that these salines present, either "singly or in combination," is as follows:

The Tertiary salt deposits of the Gulf coast and the Cretaceous salines of eastern Texas and northern Louisiana were initiated by the intrusion of molten rocks into the underlying Paleozoic sediments along lines of structural weakness. These great faults were the sites of frequent downward displacements during the subsidence and deposition of the sands, clays, and littoral marine sediments of the Mississippi Embayment region.

Hot, ascending solutions, containing calcium and magnesium carbonates, sodium chloride, carbon dioxide, with varying amounts of hydrogen sulphide, mingled with the artesian saline waters of the Cretaceous beds. These waters were forced upward to the surface by the hydrostatic head of the region, through channels that were opened by the faulting and movement over these intrusions of igneous rock.

Great deposits of travertine or calcareous sinter, similar to the deposit at Winnfield, La., were formed around the thermal springs that issued from these openings, the sinter continuing to build as long as the hydrostatic head was sufficient to maintain the flow.

If we assume, as we must, a constant hydrostatic head for the artesian saline waters of the Cretaceous beds, and if the subsidence of the Mississippi Embayment region is due, as has been thought, to the increasing weight of accumulating sediments, then the flow from these springs that determined the rate of sinter building must have been closely related to the sedimentation of this section. If, on the contrary, as has been urged, the downward movement and displacements determined the amount of sedimentation by the depression of the drainage below its base level, and the subsequent silting up of its channels and concurrent building of great flood plains, such as are being formed by the Mississippi and tributary streams at the present time, it is apparent that the same secular movements which promoted such an increased accumulation of sediment would, by relatively increasing the hydrostatic head, augment the flow of water from these springs, and so add to the rate of sinter accumulation. In other words, the balance that was maintained between the rate of sedimentation and the rate of sinter building must have followed such a causal relation.

As these great deposits of calcareous dolomitic sinters were built up,

more or less gypsum was deposited in the surrounding marshes by the oxidation of the hydrogen sulphide coming in contact with the oxygenated surface waters, the H_2SO_4 so formed converting the calcium carbonate of the spring waters into the relatively insoluble sulphate.

Contemporaneously with the building of these sinters, sands and clays were deposited around their bases. At times, owing to the sudden increased activity of these springs resulting from downward movement and relative increase of hydrostatic head, the sinter accumulation encroached upon the marsh. At other times the accumulation of sediment encroached upon the sinter.

As the sinter continued to build, coincident with the subsidence and sedimentation of the region, the same excess of carbon dioxide in the ascending waters that prevented a deposition of carbonates in the channel below, attacked and re-dissolved the bottom layers. By the periodic rapid deposition of the sinter above and its slow, constant dissolution below by the carbonated saline waters, open spaces were developed that were carried upward, in which the salt was deposited from ascending solutions that were supersaturated with saline contents by the release of pressure, as well as by the evaporative losses these waters must have sustained at the surface, as the rapid sinter accumulation checked the flow from the springs.

In this way these salt deposits several thousand feet in thickness were built up contemporaneously with the sedimentation of the region and formed under their protective covers of sinter, and so we find them to-day, where they have escaped erosion and have been preserved in their entirety.

The accumulation of sinter around hot springs is relatively so rapid that the periods devoted to active sinter building, immediately following downward displacements of a few feet, will occupy but little time—a few years at most, as contrasted with the much greater intervals of the comparative inactivity or complete repose of the springs.

Such is briefly the origin of these deposits. The great deposit of travertine at the Winnfield marble quarry, and the undoubted salt core of great thickness beneath it, are too suggestive of such an origin for its presence in the midst of lower Claiborne sands and clays to be lightly ignored. Neither should the occurrence of the leaf-bearing limestone, mentioned by Veatch, be overlooked.

The following is the log of a well at Drake's Saline:[5]

	Ft.	to Ft.
Soil, muck, pebbles, logs	0	303
Limestone (like marble at Winnfield)	303	910
Salt	910	2,320
Gypsum bed at bottom	...	2,342

[5] *Bulletin No.* 429, *U. S. Geological Survey* (1910).

The log shows travertine over 600 ft. in thickness and resting upon the top of the salt.

In northern Louisiana and eastern Texas, all the Cretaceous salines are located along bayous, branches, or creeks, the artesian flow of waters from these springs having eroded their own channels.

To illustrate the rapidity of sinter accumulation:

> "The travertine of Tuscany is deposited at the Baths of San Vignone at the rate of six inches a year, at San Filippo one foot in four months. At the latter locality it has been piled up to a depth of at least 250 feet, forming a hill a mile and a quarter long and a third of a mile broad. An illustrative illustration of the rapidity with which the travertine may be deposited, is furnished by the Eocene sinter of Sezanne, Marne. This deposit contains hollow casts of flowers which fell on the growing sinter, and were crusted over with it before they had time to wither. As the material thickened round them they decayed inside, but the hardened carbonate preserved an accurate mould of their forms. When hot wax is injected into these cavities, and the surrounding lime is dissolved away with acid, perfect casts of the flowers are obtained."[6]

There is nothing antagonistic to this hypothesis in the stratigraphy of this section of the Mississippi Embayment region, as revealed by numerous well records. Undoubted marine beds occur; and it is quite certain that salt cores with their protective caps of sinter were depressed below sea level during the upper Cretaceous and their spring waters were effectually sealed by the deposition of the marine sediments above. This probably happened to the salt deposits of eastern Texas and northern Louisiana, but could in no way have affected the tertiary salines of the Gulf Coast region, which were doubtless of a later development, although, in all essential details, analogous to the Cretaceous salines farther north.

Some of the salines of the Gulf region were submerged in late Tertiary seas. Well records show at depths of about 1,000 ft. the crystalline limestone that overlies the salt at Spindletop. Other domes, confined to a zone running practically due east and west from Beaumont, probably have had a similar history. As the axis of the folding parallels this direction a sudden downward movement of considerable magnitude no doubt depressed these salines below the Gulf level.

A microscopic examinaton of the rock salt from Avery's Island tends to corroborate and strengthen the hypothesis formulated to harmonize the field evidence gathered from a collective study of these different salt deposits.

Under the microscope, all the salt crystals examined by the writer reveal included gypsum crystals, with their ends broken or rounded. These crystals, when dissolved out of the salt, often show inclusions of foreign matter, from which the salt itself is relatively free.

The occasional presence of small quartz crystals in the Petite Anse

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[6] Geikie: *Text Book of Geology*, 4th ed., p. 476 (1903).

salt containing the same inclusions as were noted in the gypsum, seems to indicate that both were formed under similar conditions.

Since the sinter at Winnfield contains great numbers of these small crystals of quartz, showing, under a magnification of 300 diameters, the same inclusions, or what appear to be so, it is probable that they were derived from the sinter; and their presence in the salt must be ascribed to the downward migration of all insoluble material formed in the sinter by the constant deposition above and solution below. When the salt is broken, a strong odor of hydrogen sulphide is emitted.

The quaquaversal structural features of these domes are no doubt due to faulting and downward displacements. The introduction of hard cores of salt into these planes of faulting contributed greatly to such structural features as characterize these salines.

I have not attempted a detailed criticism of the different theories advanced in explanation of these unique deposits. All are of value, insomuch as the discussion has eliminated much misconception and has contributed greatly to our knowledge of these salines.

Discussion

G. D. Harris, Ithaca, N. Y. (communication to the Secretary *).—Since I have on several occasions[1] expressed at length my general views on this interesting subject, it seems fitting that I endeavor here to comment as briefly as possible on a few points suggested by the paper. Doubtless you all know of the rather extensive publications of Lachmann on mid-European salines and his scarcely successful attempt to make the phenomena presented by our domes fit into his "Exzeme" theory based on studies of European phenomena.

This paper is a most welcome indication that scientific America is very much alive to the difficulties that have been encountered in attempts already made to explain the origin of the salt domes in the lower Mississippi Embayment region. Praiseworthy, too, is the attempt to explain these phenomena by agencies still in operation. I may, perhaps, be permitted to state here that a former student of mine, and co-worker in Louisiana, A. C. Veatch, is now interested in the Algerian region, where it would seem similar dome-phenomena are to be found, and hopes may well be entertained that this shrewd and zealous geologist will some day furnish additional light on the subject from that distant land. As for myself, I have had occasion to examine many of the domes recently in great detail, and have had a student working in the laboratory along the line of

* Received Mar. 6, 1915.

[1] *Economic Geology*, vol. iv, No. 1, pp. 12 to 34 (1907).
Bulletin No. 7, *Louisiana Geological Survey*, pp. 59 to 83 (1907).
Bulletin No. 429, *U. S. Geological Survey*, pp. 6 to 12 (1912).
Popular Science Monthly, January, 1913, pp. 187 to 191.

precipitation from solution through decrease in temperature, and hence may say definitely that such precipitation, with the formation of salt crystals, is quickly and copiously obtained. The necessity of assuming surface evaporation in the formation of these crystalline, saline masses seems therefore to be no longer pressing. The force exerted by growing salt crystals is next to be taken up.

I will limit my remarks to two points in the theory before us. First, the assumption of volcanic activity. Having made complete trigonometrical and magnetic surveys about two representative domes where salt masses are known to occur I have found that the local deflections observable are within the limits of instrumental error, about 2 min. The marked deflections about the nearest igneous matter, the peridotites of southwest Arkansas, I have personally measured; those about Hot Springs are too well known to need commenting on here. Local-heated nuclei or even cold plutonic plugs would almost certainly have made themselves known, if present, by these investigations. The "hot" waters so far observed in any of the great flows from wells put down in domes or off of domes have revealed scarcely an increase of 1° F. to every 50 ft. of descent. I have seen nothing suggesting local hot rocks below. Again, the use of such terms as travertine and "calcareous sinter" for the hard blue and white highly crystalline limestone (called "marble" locally) I believe is apt to produce a wrong impression. That there may be a slight solution and occasional redeposition about the peripheries of these saline domes no one can doubt, though the redeposition of chloride of sodium in present climatic conditions, so far as a permanent rock-forming process is concerned, is entirely out of the question. The rocks referred to as light-gray limestone at Drake's and elsewhere are more arenaceous than calcareous, as determined by subsequent examination, and are of the ordinary sedimentary type, having nothing in common with dome rock, as intimated in this paper. The fact that they contain leaves is therefore not strange, nor does it imply contemporaneity of dome formation and sedimentation. Personally I would be as sanguine of finding fossils in any batholith as in any dome rock.

Second, as to quaquaversal structure. This to me is the key to the whole situation. And it is with distinct regret that I find this all-important subject dismissed with no new field observations and with general remarks covering less than four lines. Naturally one might believe *a priori* that after a dome mass had formed, there might be some slight settling down about the same, giving rise to a certain amount of dragging and bending of the edges of bedding planes against the dome mass. But field observations will allow of being treated with no such academic and slight consideration. How can "faulting and downward displacements" explain the presence of Cretaceous beds lying upon and flanking a local, small, circular dome that peers out through Lower Claiborne deposits

2,000 ft. above the undisturbed Cretaceous deposits? How may they be so crushed, compressed, and hardened till all bedding planes are obliterated and systems of cleavage or joint planes are established by simply "faulting downward"? Perhaps I can make the difference between the author's and my views on this subject of relative motion clear by means of a homely comparison. Suppose on entering your newly completed house you notice the carpenters have left the point of a nail protruding through the floor. Being of a scientific turn of mind you may investigate the matter closely, and finding the woody matter broken upward all about the nail in quaquaversal manner, conclude either that some careless "hand" had driven the same up from below, or you could possibly imagine that the nail had remained *in situ* and the whole floor had sunk about it. So, too, you may imagine the whole rock floor of northwest Louisiana as having sunk 2,000 ft. about these local circular salt spikes, but I still prefer to think of the latter as having been driven up from below according to the theory discussed under the references herewith given.

Barite of the Appalachian States

BY THOMAS L. WATSON AND J. SHARSHALL GRASTY,* CHARLOTTESVILLE, VA.

(New York Meeting, February, 1915)

Introduction

The users of barite in the United States derive their supply partly from the domestic production and partly from the imports from foreign countries. According to the Mineral Resource division of the U. S. Geological Survey, the domestic production of barite in 1913 was 45,298 short tons, valued at $156,275, or $3.45 a ton, and the imports of crude barite were 35,840 short tons, valued at $61,409, or $1.71 a ton. These figures clearly show a decided advantage in cost of the foreign over the domestic barite to users in the United States. The difference in cost is the only advantage to the consumer of foreign barite, for there are large reserves of the mineral in the United States of good grade. The production of many of the operating mines is capable of decided increase, and it is believed that the operating mines together with the large reserves of undeveloped deposits of barite are entirely adequate to meet the demands of the users in the United States.

Deposits of barite are known in California, Idaho, Nevada, and Alaska, but most of them are undeveloped because there is, apparently, no market for the material in that region. The domestic production is derived from Missouri and the Appalachian States, the greater part being obtained from Missouri.

Following the outbreak of war in Europe many users of foreign barite in the United States have been forced to seek their supplies at home. It seems opportune at this time, therefore, that a general review be given of the barite industry in the Appalachian States, one of the two areas from which the domestic supply of the mineral is derived, and that attention be directed to the undeveloped deposits in the region. Briefly, then, this paper reviews the barite industry in the Appalachian States, and describes the occurrence, preparation, and uses of the mineral.

General Character

Barite, known commercially as barytes or heavy spar, is barium sulphate ($BaSO_4$), containing when pure BaO, 65.7 per cent., and SO_3, 34.3 per cent. The specific gravity is 4.3 to 4.6, and the hardness 2.5

* University of Virginia.

to 3.5. The hardness is about that of calcite, but barite can be readily distinguished from calcite by its greater specific gravity and by not effervescing with acids. The mineral is usually white, but it is often found stained red, pink, or yellow by iron oxide. It is usually crystalline, opaque to translucent, and occurs in a variety of forms, commonly as aggregates of straight and slightly curved cleavable plates, as granular, fibrous, and earthy masses, and as single and clustered crystals. It is rarely found pure in workable deposits, the common impurities being silica, lime, magnesia, and the oxides of iron and aluminum. In many deposits galena and fluorite are associated with the mineral. The minable material will usually carry from 95 to 98 per cent. barium sulphate.

Production[1]

During the 10-year period 1904 to 1913, inclusive, the production of barite has been entirely from the following States, arranged alphabetically and without reference to order of production: Alabama, Georgia, Kentucky, Missouri, North Carolina, South Carolina, Tennessee, and Virginia. Not all of these, however, have produced during the same year. In 1913, the production came from Missouri, North Carolina, Virginia, Georgia, and South Carolina. The Kentucky deposits did not produce in 1912 or 1913.

The following table gives the total production of crude barite in the Appalachian States and in the United States in short tons from 1904 to 1913, inclusive:

Production of Barite

	Total Production,[a] Appalachian States		Total Production,[b] United States	
	Quantity	Value	Quantity	Value
1904	40,229	$99,406	65,727	$174,958
1905	21,474	64,705	48,235	148,803
1906	21,362	66,888	50,231	160,367
1907	45,582	129,318	89,621	291,777
1908	22,208	63,674	38,527	120,442
1909	27,130	89,919	61,945	209,737
1910	19,997	46,148	42,975	121,746
1911	16,945	41,412	38,445	122,792
1912	12,948	36,278	37,478	153,313
1913	14,167	38,637	45,298	156,275

[a] Includes Virginia, North Carolina, South Carolina, Georgia, Tennessee, Kentucky, and Alabama.

[b] Includes all States under (*a*) and Missouri.

[1] Unless otherwise stated all statistical data given in this paper are taken from *Mineral Resources of the United States*, U. S. Geological Survey.

Average Price per Ton

	Missouri	North Carolina	Tennessee	Virginia	Other States	Total
1905..	$3.14	$3.90	$1.62	$4.30	....	$3.08
1906..	3.24	....	1.67	3.85	$2.94[a]	3.19
1907..	3.09	3.76	1.78	3.55	4.18[a]	3.26
1908..	3.48	....	1.43	....	3.51[b]	3.13
1909..	3.44	....	....	....	3.31[c]	3.39
1910..	3.29	....	1.54	....	2.55[d]	2.83
1911..	3.79	....	2.27[c]	....	2.63[f]	3.19
1912..	4.77	....	2.34[e]	....	2.99[g]	4.09
1913..	3.78	....	1.70	....	2.91[h]	3.45

[a] Includes Alabama, Kentucky, Georgia.
[b] Includes Georgia, North Carolina, and Virginia; Kentucky was $4.11.
[c] Includes Georgia, Kentucky, North Carolina, Tennessee, and Virginia.
[d] Includes Georgia, Kentucky, North Carolina, and Virginia.
[e] Includes Tennessee and Kentucky.
[f] Includes Georgia, North Carolina, South Carolina, and Virginia.
[g] Includes Georgia, North Carolina, and Virginia.
[h] Includes Georgia, North Carolina, South Carolina, and Virginia.

Imports

The subjoined table gives the imports of barite for consumption during the last 10 years, from 1904 to 1913, inclusive.

Barite Imported into the United States, 1904 *to* 1913, *in Short Tons*

	Manufactured		Unmanufactured		Total Value
	Quantity	Value	Quantity	Value	
1904.............	6,630	$48,658	7,492	$27,363	$76,021
1905.............	4,803	39,803	14,256	62,459	101,262
1906.............	4,807	37,296	9,190	27,584	64,880
1907.............	11,207	96,542	20,544	76,883	173,425
1908.............	3,401	29,168	13,661	58,822	87,990
1909.............	3,016	25,679	11,647	29,028	54,707
1910.............	3,565	29,782	21,270	48,457	78,249
1911.............	3,147	22,083	20,214	36,643	58,726
1912.............	3,679	26,848	26,186	52,467	79,315
1913.............	5,463	38,155	35,840	61,409	99,564

The total yearly value of the imports of barium compounds, 1904 to 1913, including barium carbonate (witherite), barium binoxide, barium chloride, and blanc fixe or artificial barium sulphate, follows:

Total Value of Imports of Barium Compounds, 1904 *to* 1913

Year	Value
1904	$242,804
1905	257,427
1906	335,011
1907	357,117
1908	319,114
1909	399,376
1910	470,449
1911	398,213
1912	376,017
1913	391,470

The mineral witherite (barium carbonate) is admitted free of duty but under the Underwood tariff the manufactured barium carbonate is dutiable at 15 per cent. *ad valorem*, the natural barite at 15 per cent. *ad valorem*, the artificial barium sulphate or blanc fixe at 20 per cent. *ad valorem*, barium binoxide at 1½ c. a pound, and barium chloride at ¼ c. a pound. On raw barite the duty is $1.50 per ton, on prepared or manufactured barite $5.25 per ton, and on blanc fixe or artificially prepared barium sulphate the duty is ½ c. per pound.

Geographic and Geologic Distribution

Barite has rather wide geographic distribution in the Appalachian States and occurrences of the mineral are known in each of the major physiographic divisions of the region: (1) The Piedmont Plateau, (2) the Appalachian Mountains, (3) the Appalachian Valley, (4) the Cumberland Plateaus, and (5) the Interior Lowlands. (See map, Fig. 1.) It is found in workable quantity in many localities within the Appalachian States, but in recent years the sources of supply have been from Virginia, North Carolina, South Carolina, Georgia, Tennessee, and Kentucky. Deposits of the mineral are also known in Alabama, Maryland, and Pennsylvania. Practically the entire domestic production of the mineral comes from Missouri and the Appalachian States, the former supplying about 65 per cent. of the total production.

Geologically, the barite deposits of the Appalachian States are associated with rocks which range from pre-Cambrian to Triassic in age, but the exact age of the deposits themselves is unknown.

The age of the rocks in which the barite deposits are found may be tabulated as follows:

1. Triassic: Virginia.
2. Carboniferous (Mississippian): Western Kentucky.
3. Cambro-Ordovician: Central Kentucky, Tennessee, Appalachian Valley region of Virginia, Georgia. Alabama, Maryland, and Pennsylvania.
4. Pre-Cambrian and Cambrian: Piedmont Plateau province of Virginia, North Carolina, including the Madison County deposits in the western part of the State, and South Carolina.

Of these four horizons, the Cambro-Ordovician is vastly the most important in the Appalachian States and to it belong the deposits of Missouri, the principal producing area in the United States. The deposits of barite concentrated in the residual clays derived by weathering from the rocks of the several horizons tabulated above are later in age, probably much later.

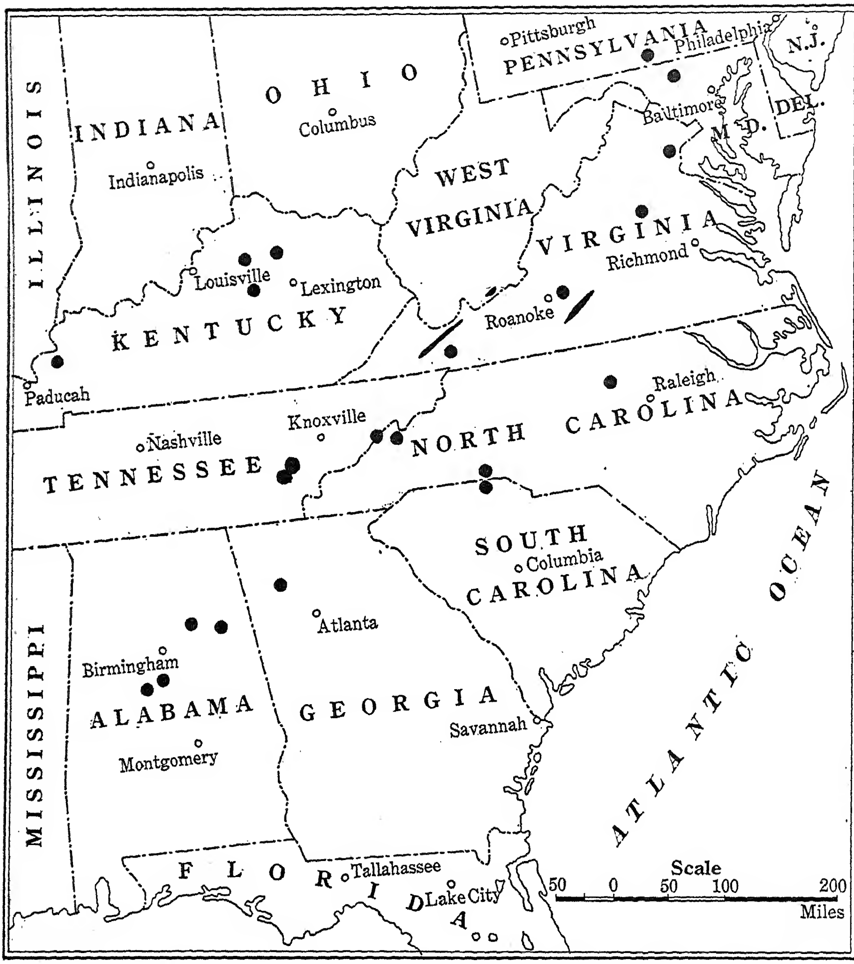

FIG. 1.—MAP SHOWING BARITE DEPOSITS IN THE APPALACHIAN STATES.

MODE OF OCCURRENCE

Barite is not a pyrogenetic mineral, nor is it known in contact-metamorphic deposits, in pegmatite veins or dikes, or as a product of dynamo-regional metamorphism. It is a common mineral, however, in many kinds of rocks—igneous, sedimentary, and metamorphic; and in nearly all

if not in all cases it has formed by deposition from aqueous solutions. Many writers record the deposition of barite from spring and mine waters of carbonate and even sulphate composition, but especially sodium chloride waters.[2] Depositions from both hot and cold water are recorded.

Thus far, therefore, the occurrence of barite is limited to deposits of moderate and shallow depth, either nearby, sometimes in, or removed from igneous rocks, but not formed, according to Emmons,[3] in deposits of the deep-vein zone.

The occurrences of barite in deposits of sufficient concentration to be of commercial value are (1) as veins, (2) as replacements of limestone, and (3) as a residual mineral in clays resulting from weathering.

Vein occurrences of the mineral may include (1) barite as a common gangue mineral in veins of metallic ores; or (2) barite as the chief mineral in veins, with little or essentially no development of metallic ores, in sedimentary rocks (sandstones and limestones), in metamorphic crystalline rocks (schists and gneisses), and in igneous rocks. Well-known examples of each of these occurrences of the mineral in the southern Appalachian States are described in another part of this paper. Likewise, the occurrence of barite as a replacement of limestone finds ample illustration in Virginia, where it forms an important type of deposit.

Finally, an important mode of occurrence of barite in the known commercial areas of the United States is as a residual mineral, forming masses of varying sizes and shapes in the clays derived by weathering from the decay of limestones. A large part of the barite mined in the Appalachian States and in Missouri is taken from the residual mode of occurrence of the mineral.

Character of Associated Rocks

From the mode of occurrence of barite discussed above, the mineral shows wide range as to kind of rock in which it is found. The numerous occurrences of barite in the Appalachian States afford ample illustration of the wide range in the kind of rock with which it is associated. These include rocks of each of the three major groups—sedimentary, igneous, and metamorphic—and workable concentrations of the mineral are known in each of the three rock groups. Of these, limestones and their residual clays among the sedimentary rocks are vastly the most important sources of supply of the mineral.

Of the sedimentary rocks, limestones and sandstones or their residual clays, especially limestone, are the usual ones which yield workable de-

[2] Lindgren, W.: *Mineral Deposits*, pp. 98 to 102 (1913). References to the literature cited.

Clarke, F. W.: *Bulletin No.* 491, *U. S. Geological Survey*, pp. 553 to 555 (1911). References cited.

[3] Emmons, W. H.: *Economic Geology*, vol. iii. No. 7, p. 618 (Oct.-Nov., 1908).

posits of the mineral. In at least one locality, however, Prince William County, Virginia, barite is associated with the Triassic red shales and impure limestones as veins. Occurrences of barite in limestone or its residual clays are noted in each of the larger physiographic divisions of the Appalachian States.

The known deposits of barite in igneous and metamorphic siliceous crystalline rocks in the Appalachian region are limited to the Piedmont Plateau, where the mineral occurs as veins and replacements, chiefly in schists and crystalline limestones, and occasionally in granite. Examples are known in Virginia, North Carolina, and South Carolina.

Associated Minerals

These will vary for the individual deposit, but the more commonly occurring ones are found to be dependent in large measure upon the mode of occurrence of the barite. In the vein and replacement types of deposits[4] barite is often associated with metallic sulphides, especially galena, sometimes sphalerite, chalcopyrite, and pyrite; and several of these may be developed in the same deposit. When in quantity, galena is harmful, since it renders the ground product too dark. Quartz, fluorite, and calcite are frequent accompaniments of barite in the deposits of some districts. Fluorite is especially noted in the Kentucky and Tennessee vein occurrences of barite, and in some of the Valley occurrences in Virginia.

In the residual occurrences of barite, the associated minerals will naturally vary with the character of the original rock, but probably the most frequently occurring ones, exclusive of the clay through which the nodular barite is scattered, are the oxides of iron and manganese, and silica in the form of both quartz and chert. In addition to these, tremolite and biotite are noted in some of the Virginia occurrences of barite in the crystalline rocks.

Occurrence and Industry by States

In the following pages the barite industry in the Appalachian States is summarized by States. The treatment is by alphabetical arrangement of the States regardless of their rank as producers.

Alabama

The known barite deposits in Alabama are confined to the following five counties of the Paleozoic area in the central and northeastern parts

[4] No account is here taken of the veins of metallic ores in which barite occurs as a gangue mineral, nor of the occurrences of ore deposits in which barite is developed as a subordinate mineral.

of the State: Bibb, Calhoun, Etowah, St. Clair, and Shelby. The most important localities include (1) Maguire Shoals on Little Cahaba River, at the "Sinks" on Six Mile Creek, and near Pratt's Ferry in Bibb County; (2) near Tampa in Calhoun County; and (3) near Greensport in St. Clair County. Barite also occurs at other places in these counties, and is reported from two localities in Shelby County, but these appear to be of doubtful commercial value.[5]

The Alabama deposits of barite are residual in type, probably concentrated in the clay from veins and replacements in the Cambrian and Ordovician limestones, as the early literature contains references to vein occurrences of the mineral. The mineral occurs in loose pieces on the surface, and in lumps, nodules, and irregular masses imbedded in the residual clays derived from the Knox dolomite of Cambrian and Ordovician age, and the Pelham or Trenton (Chickamauga) limestone of Ordovician age. The deposits seem to be localized along lines of folding and faulting, the more important ones being found near the contact of the Knox dolomite with the Pelham or Trenton limestone, although some of the smaller deposits occur entirely within the limits of the residual clays of the Knox dolomite.

In some of the earlier reports of the State Survey, Dr. Eugene A. Smith, State Geologist, refers to many occurrences of barite in veins. Concerning the occurrence of barite in Bibb County, Smith says:[6] "Barite, or heavy spar, is of frequent occurrence in the Quebec [Knox] dolomite. Maguire's Shoal on the Little Cahaba, and the 'Sinks' on Six Mile Creek, may be mentioned as localities, but barite is found in veins, in many other places." Again, the same author says of barite in Shelby County:[7] "Just south of Calera, and associated with beds of limestone of this group [Chazy and Trenton], is a small bed of limonite exposed in the dry channel of a little branch. The limonite has very intimately mixed with it, in varying proportions, *barite* or *heavy spar*. Some portions of the bed showed almost pure limonite, others limonite and barite in equal proportions, and still others, almost pure barite. Scarcely a hand specimen could be obtained without both minerals.

"Barite in veins constantly accompanies the belt of Chazy limestone described above."

In Berney's *Handbook of Alabama*,[8] Smith states that barite is found in veins in many localities in the Knox dolomite.

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[5] Smith, E. A., and McCalley, H.: *Index to the Mineral Resources of Alabama, Geological Survey of Alabama*, pp. 62, 63 (1904).

Grasty, J. S.: Barite Deposits of Alabama, *The Tradesman*, pp. 35, 36 (July 17, 1913).

[6] Smith, E. A.: *Report of Progress for* 1875, *Geological Survey of Alabama*, p. 95 (1876).

[7] *Idem*, p. 118.

[8] Berney, Saffold: *Handbook of Alabama: A Complete Index to the State;* with a geological map, etc., p. 156 (Mobile, Ala., 1878).

The mineral is generally of good quality, usually very pure and white, sometimes grayish and bluish in color, and stained with iron oxide. In some cases the barite is associated with or occurs near limonite deposits, and at times carries more or less galena. Notwithstanding the good quality of the barite in most of the important localities, developments have thus far been slight, and so far as known little or none of the mineral has been placed upon the market. There has been no production of barite in Alabama since 1906. Probably the chief cause for the lack of development of some of the better deposits of the mineral is to be attributed to the absence of a plant in the State to consume the output. The distance and hence freight rates to barite mills in adjoining States are prohibitive for the Alabama product because of a nearer and cheaper source of supply of the mineral.

Georgia

Barite has been mined at Elton, Murray County, and in the Cartersville district, Bartow County, in northwest Georgia. For some years,

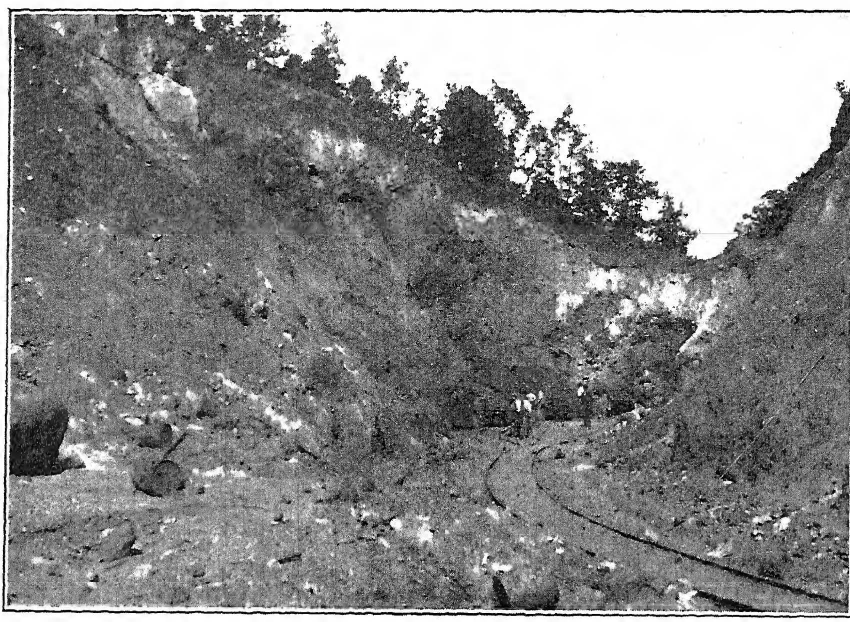

Fig. 2.—Barite Mine Near Cartersville, Bartow County, Georgia. (S. W. McCallie, Photo.)

however, the entire production has come from the Cartersville district where iron ore, ocher, and barite are mined. (Fig. 2.)

The Elton deposit in Murray County has not been worked in recent years. The composition[9] of the barite from this locality is: $BaSO_4$,

[9] *The Mineral Industry*, vol. xix, p. 69 (1910).

98.82; Al_2O_3, 0.2; Fe_2O_3, 0.33; SiO_2, 0.27; moisture, 0.38 per cent. The mineral is reported to be lower in iron than that mined in the Cartersville district.

The Cartersville district, occupying the southeastern half of Bartow County, includes about 70 square miles, nearly equally divided between the Paleozoic formations (quartzite, limestone, shale, and dolomite) on the west and the older crystalline and metamorphic rocks (schists, gneisses, slates, and conglomerates) of the Piedmont Plain and Appalachian Mountains on the east.[10]

The barite deposits are associated with certain of the Paleozoic formations which occupy the west half of the district, and which named in descending order are: (1) Knox dolomite, (2) Conasauga and Rome formations, (3) Beaver limestone, and (4) Weisner quartzite.

Except the Knox dolomite, these formations belong to the middle or lower Cambrian, and it is probable that the lower portion of the Knox should be classed with the Cambrian. The barite deposits are associated chiefly with the Weisner quartzite and Beaver limestone, though some have been found with the Knox dolomite.

The Weisner quartzite, the basal member of the Cambrian, forms a nearly continuous belt about 19 miles long with an average width of from 1 to 2 miles. It has an estimated thickness of 2,000 to 3,000 ft., and consists of fine-grained quartzite, some beds of fine conglomerate, and interbedded siliceous shale. The conglomerate beds frequently show angular fragments of feldspar, indicating that a part of the material composing it was derived from the granite area to the east. The formation has been intensely folded, and is probably cut by numerous faults, as

[10] For a description of the geology and ore deposits of the Cartersville district, see:

Spencer, J. W.: The Paleozoic Group, *Geological Survey of Georgia*, pp. 99 to 107 (1893).

Hayes, C. W.: Geological Relations of the Iron-Ore Deposits in the Cartersville District, Georgia, *Trans.*, xxx, 403 to 419 (1900).

Hayes, C. W., and Phalen, W. C.: A Commercial Occurrence of Barite near Cartersville, Ga., *Bulletin No.* 340, *U. S. Geological Survey*, pp. 458 to 462 (1908).

McCallie, S. W.: A Preliminary Report on a Part of the Iron Ores of Georgia, *Bulletin No.* 10-*A*, *Geological Survey of Georgia*, pp. 109 to 176 (1900).

Watson, Thomas L.: Geological Relations of the Manganese-Ore-Deposits of Georgia, *Trans.*, xxxiv, 207 to 253 (1903).

——— A Preliminary Report on the Manganese Deposits of Georgia, *Bulletin No.* 14, *Geological Survey of Georgia*, pp. 32 to 99 (1908).

——— The Yellow Ocher Deposits of the Cartersville District, Bartow County, Georgia, *Trans.*, xxxiv, 643 to 666 (1903).

——— A Preliminary Report on the Ocher Deposits of Georgia, *Bulletin No.* 13, *Geological Survey of Georgia* (1906). 81 pp.

Maynard, T. P.: A Report on the Limestones and Cement Materials of North Georgia, *Bulletin No.* 27, *Geological Survey of Georgia*, (1912). 293 pp.

Watson, T. L., and Grasty, J. S.: The Geology and Barite Deposits of the Cartersville, Georgia, District, *The Tradesman*, May 22, 1913, pp. 35 to 37.

evidenced by its crushed and brecciated condition in many places. So extensive is the crushing and brecciation in some of the larger mine openings in the district that the original bedding of the quartzite is entirely obscured. The resulting structural conditions of the quartzite have been particularly favorable to chemical action and are responsible for the deposition of the mineral deposits, especially ocher and barite, contained in it.

A chemical analysis made by the N. P. Pratt laboratory on specimens collected by the senior writer from different exposures of the formation gave 4.46 per cent. of barium sulphate.

Stratigraphically above, but topographically below the Weisner quartzite, is a thickness of 800 to 1,200 ft. of a gray crystalline magnesian limestone, which is shaly in places and at times chert-bearing, known as the Beaver limestone. This limestone forms a narrow belt of lowland along the western margin of and parallel to the ridges of the harder Weisner quartzite. Outcrops of the limestone are seldom seen and its surface is covered by a deep mantle of residual red clay by which the formation may be readily traced.

Barite has two modes of occurrence in the Cartersville district: (1) As deposits from solution in fractures and cavities in the Weisner quartzite in intimate association with the deposits of yellow ocher, and (2) as crystalline nodules and masses of varying size imbedded in the residual clays derived from the decay of the Weisner quartzite and Beaver limestone. Each of these has proved to be a source of supply of barite, but the principal commercial source of the mineral has been from the residual clays. In addition to yellow ocher, the barite deposits of the Cartersville district are more or less closely associated with iron ore.

Barite is found in nearly all the ocher deposits of the Cartersville district. It is known to the miners as the "flowers of ocher," and since it remains in the residual soils derived from the quartzite, it affords the best means of tracing the ocher deposits. It has been shown by Hayes, and later confirmed by Watson, that the ocher deposits with which barite is associated are the direct replacements of the silica in the Weisner quartzite from thermal solutions circulating along the zones of fracture. These relations are best indicated at the mine of the Georgia Peruvian Ocher Co., at the wooden bridge over Etowah River. Here large clusters and groups of magnificent barite crystals are found in the pockets of ocher and in the fractures and cavities of the quartzite. The usual occurrence of barite is in divergent groups of massive tabular crystals, giving a crested appearance and grading into both straight and curved laminated masses. Groups of crystals weighing 25 lb. and more are not uncommon. It seems probable, as Hayes states, that the crystals of barite "were probably deposited after the conditions favorable for the solution of silica and the deposition of ocher had passed. Groups of acicular

crystals of this mineral, several inches in length, are not uncommon. It also occurs in white granular veins."[11]

A second and commercially more important occurrence of barite in the Cartersville district is as nodules and masses of varying sizes and shapes imbedded in the residual clays. Deposits of this type of the mineral are rather widely distributed over the district, but probably many of them will not prove profitable because of lack of sufficient concentration for mining alone. However, a number of promising deposits have been opened, but further development is necessary before a definite statement can be made as to the extent, quality, and workability of the deposits in the district.

The principal barite deposits occur near the south end of the quartzite belt where the structure has been shown to be anticlinal. The deposits are found on the east side of the anticline lower down the slope but adja-

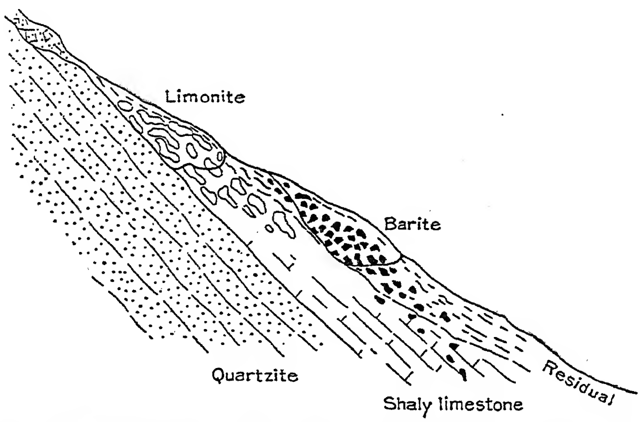

FIG. 3.—SKETCH SECTIONH SOWING RELATIONS OF BARITE AND LIMONITE TO UNDERLYING FORMATION NEAR CARTERVILLE, GA. (AFTER HAYES AND PHALEN.)

cent to a belt of limonite which rests directly on the quartzite and has been mined for many years. According to Hayes and Phalen,[12] "while this belt has not been sufficiently prospected to prove the continuity of the deposits, their thickness is indicated by a tunnel which has been driven into the side of the hill about 300 ft. north of the present workings. This tunnel penetrates 120 ft. of the barite-bearing clay and then enters the iron ore." (See Fig. 3.)

Concerning the relations of the limonite and barite to the underlying formations Hayes and Phalen say:[13] "One striking feature is the sharp separation between the two minerals; each appears to occupy a definite horizon, and so far as observed there is no intermingling. While both in

[11] Hayes, C. W.: *Trans.*, xxx, p. 418 (1900).

[12] Hayes, C. W., and Phalen, W. C.: *Bulletin No.* 340, *U. S. Geological Survey*, p. 461 (1908).

[13] *Idem*, p. 461.

their present condition are residual, they appear originally to have replaced distinct beds in the shaly limestone overlying the quartzite, the iron being deposited immediately adjacent to the quartzite, and the barite, for the most part, at least, in higher beds. It is highly probable that deposition of the barite and limonite took place, in part, at least, simultaneously. It is quite certain that gravity has been largely instrumental in concentrating the former into a workable deposit."

The barite deposit is described as having a thickness of 50 ft. normal to the surface slope; is intermingled with residual clays containing fragments of quartzite; and the barite makes up about a third of the material mined. The barite forms nodules weighing several ounces up to boulders of several hundred pounds; is mostly massive and compact granular in texture; and of pure white to bluish color.

Kentucky[14]

Barite is reported to have been first marketed from Kentucky in 1903, when shipments were made from western Kentucky and from Scott County in central Kentucky. With the exception of a small tonnage in 1907 from western Kentucky, the production has been from the central part of the State.

In Kentucky barite deposits are known in (1) the central and (2) the western parts of the State. Thus far mining of barite in the State has been confined chiefly to the central or Blue Grass region, which comprises the following 16 counties: Anderson, Bourbon, Boyle, Clark, Fayette, Franklin, Garrard, Harrison, Henry, Jessamine, Lincoln, Madison, Mercer, Owen, Scott, and Woodford. Of these, Boyle, Fayette, and Garrard counties, in the order named, are the principal producers of barite. The map, Fig. 4, shows the location of the principal barite deposits in the central Kentucky region.

The barite deposits of central Kentucky are confined to the Ordovician and are most prominently developed in the lower or Mohawkian division of the Ordovician, more especially in the limestones of Stones River and Trenton age and in the Eden shale. The rocks are all sedimentary and

[14] Fohs, F. Julius: Barytes Deposits of Kentucky, *Kentucky Geological Survey*, ser. iv, vol. i, pt. 1, pp. 441 to 588 (1913).

Grasty, J. S.: Barite Deposits of Kentucky, *The Tradesman*, July 3, 1913, pp. 33, 34.

Miller, A. M.: The Lead and Zinc Bearing Rocks of Central Kentucky, with Notes on the Mineral Veins, *Bulletin No.* 2, *Kentucky Geological Survey* (1905), 35 pp.

Ulrich, E. O., and Smith, W. S. T.: Lead, Zinc, and Fluorspar Deposits of Western Kentucky, *Bulletin No.* 213, *U. S. Geological Survey*, pp. 205 to 213 (1903).

——— The Lead, Zinc, and Fluorspar Deposits of Western Kentucky, *Professional Paper No.* 36, *U. S. Geological Survey* (1905), 218 pp.

The Barite Deposits of Western Kentucky and Union County will be described in *Bulletin No.* 15, to be issued shortly by the State Geological Survey.

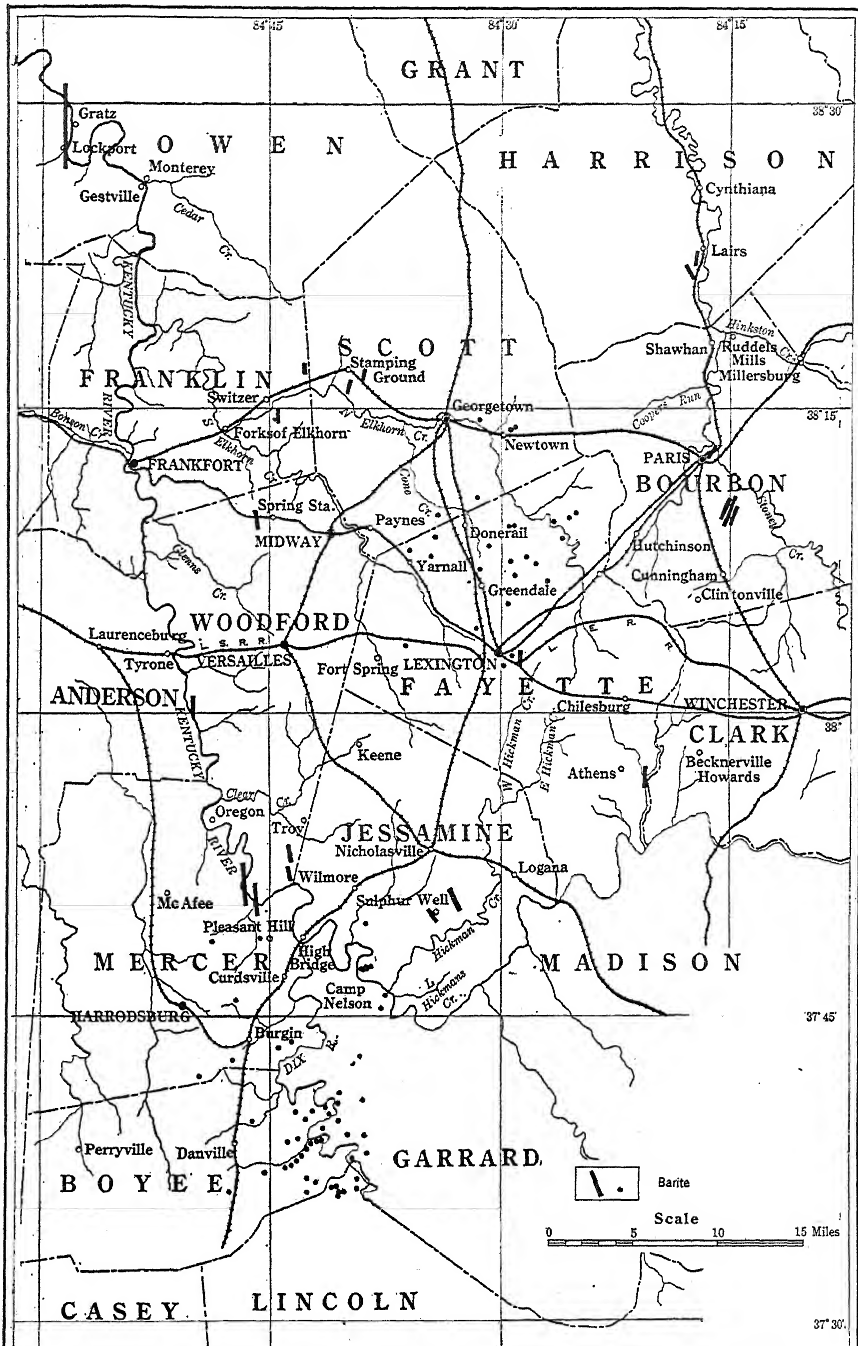

Fig. 4.—Principal Barite Deposits in Central Kentucky. Barite Shown by Solid Black Dots and Dashes. (Adapted From Published Reports of Kentucky Geological Survey.)

the important deposits of barite are confined to the limestone members. The barite occurs chiefly in veins filling simple fissures or faults, chiefly the latter. The veins are vertical or slightly inclined. They vary in width up to 24 ft., but the usual width is from 1 to 3 ft. The thickness of individual veins varies much from point to point, swelling or pinching both laterally and vertically. They are apt to become lean and barren in the argillaceous beds, but are of good grade in the limestone beds. According to Miller, the veins seem to have considerable vertical extent, and mining operations on some of them have extended to depths ranging from 100 to 325 ft., without noticeable tendency to close up.[15]

The barite deposits of central Kentucky are either of the fissure, breccia, or replacement type, or a combination of these. The breccia deposits are associated with faults. The principal minerals associated with the barite are calcite, fluorspar, sphalerite, and galena, with quartz, marcasite, pyrite, chalcopyrite, smithsonite, limonite, and manganese oxide less frequent. Strontianite and celestite are reported to occur (see analyses below).

The following analyses[16] of barite from the central Kentucky region are of interest:

Analyses of Kentucky Barite

	I	II	III	IV	V	VI	VII	VIII	IX	X	XI
Ignition	0.17	0.34	0.87	0.28	0.62	0.23	1.40	0.65	0.15	1.19	0.28
SiO_2	0.34	0.11	0.11	0.25	0.10	0.05	0.55	0.18	0.10	0.20	0.23
Fe_2O_3	0.06	0.02	0.04	0.06	0.06	0.03	0.06	Trace	0.04	0.08	Trace
BaO	50.86	50.96	60.26	64.72	59.80	50.96	54.95	10.71	64.68	49.98	60.55
SrO	8.99	11.65	3 80	Trace	2.41	11.10	6.75	45.06	0.20	11.07	3.72
CaO	3.61	Trace	0.76	Trace	2.09	0.88	0.20	0.25	Trace	Trace	Trace
SO_3	33.87	36.31	34.60	33.82	33.23	35.66	34.22	42.28	33.97	35.12	35.38
F	2.45	Trace	0.52	0.00	1.42	0.60	0.14	0.00	0.00	0.50	0.00
ZnO	Trace	0.00	0.00	0.00	0.00	0.00	1.23	0.00	0.00	1.50	0.00
Total	100.35	99.39	100.96	99.13	99.73	99.51	99.50	99.13	99.14	99.68	100.16

I. John Baughman prospect, one-half mile north of Danville, Boyle County.
II. Lee River prospect, one-half mile south of Harrodsburg, Mercer County.
III. Farris prospect, 4 miles northeast of Danville, Boyle County.
IV. Meyers prospect, near Mexico, Crittenden County.
V. Ray mine, 2 miles from Fredonia, Caldwell County.
VI. Ray mine, 2 miles from Fredonia, Caldwell County.
VII. Hayden open cut, 1½ miles south of Danville, Boyle County.
VIII. Celestite from Truesdale, Owen County.
IX. Perkins prospect, 1½ miles southeast of Burgin, Mercer County.
X. Mosby prospect, 1½ miles south of Shryock Ferry, Woodford County.
XI. Shelton open cut, 4 miles east of Danville, Boyle County.

[15] According to Fohs, the veins "may reach a depth of over 1,200 ft., a length of four or five miles and a width of 16 ft. The average present depth of the known deposits is 600 ft., at least half of which carries a commercial product."

[16] Fohs, F. Julius: *Kentucky Geological Survey*, ser. iv, vol. i, pt. 1, pp. 449 to 451 (1913).

Attention is especially directed to the large but variable amount of strontium oxide reported in most of the analyses.

In the western Kentucky region barite is found in the following eight counties: Caldwell, Christian, Crittenden, Cumberland, Livingston, Lyon, Russell, and Trigg, but the most important deposits are confined to three of these, namely, Caldwell, Crittenden, and Livingston. This area forms the well-known fluorspar district of western Kentucky, and thus far the barite deposits are incidental to those of fluorspar, just as fluorspar is incidental to the barite deposits of central Kentucky. Most of the deposits have been worked for fluorite and the sulphides, since barite is not particularly abundant. According to Fohs, the most important deposits of barite in the western Kentucky region are: "Commercial shaft, Meyers, Sullenger, and Bibb in Crittenden County; the Lindley, Stone, and Bradshaw in Livingston County; and the Satterfield, Ray, and Lowery in Caldwell County."

The barite of western Kentucky is found chiefly as a vein mineral in the fluorspar deposits, which, according to Smith, occur as (1) vein deposits filling fissures due to faulting; (2) ores cementing breccias; and (3) metasomatic replacements. The vein deposits (fluorite, barite, some galena, and some sphalerite) filling faults are by far the most important occurrence. The faults strike northeast, northwest, and east, though a few unimportant ones have a northerly direction. The veins occur in the formations of the Chester and Meramec groups, especially in Ste. Genevieve and St. Louis limestone, of Carboniferous (Mississippian) age. In Crittenden County vertical dikes and horizontal sheets of peridotite occur ranging up to 25 ft. in width. The veins dip at a high angle, usually 60° or more, and are subject to change both in dip and strike. They show considerable variations in width, but the most important ones do not exceed 6 or 8 ft., and many of them are reported to pinch out completely for short distances. Besides angular fragments of the country rock, the veins are composed of the minerals fluorite, calcite, barite, sphalerite, and galena, occasionally quartz and pyrite, and rarely others. Banding is a pronounced structural feature.

Concerning the occurrence of barite, Smith says:[17]

"Barite is common in some of the veins, although in many if not most of them it is not found at all. Taking the district as a whole, however, it is probably next in abundance to calcite as a vein mineral. In many cases where it occurs it forms only a small proportion of the vein, although occasionally it is the chief constituent, as at the Lowery mine, at one of the Myers prospects, and at the Bateman prospect. Seams in brecciated rocks and narrow veins sometimes consist wholly of this mineral.

"Barite occurs in all of the rocks in which fluorite is found—in Chester quartzite and in both Ste. Genevieve and St. Louis limestone. Where seen only in small amounts it was, as a rule, along the margin of the vein and never distributed through it. When occurring with fluorite in seams, it sometimes shows symmetrical banding.

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[17] *Professional Paper No.* 36, *U. S. Geological Survey*, p. 135 (1905).

"The only vein of this mineral which has been developed to any extent is that on which the Ray and Lowery shafts are situated. Here the vein has a width of several feet near the surface, pinching to from 6 inches to a foot at a depth of 30 or 35 feet. It consists almost entirely of barite, with a very small proportion of fluorite and calcite. It is more or less banded, and locally one of the central bands, from a fraction of an inch to 2 inches in width, contains a considerable amount of medium-grained sphalerite."

The various writers who have studied the western Kentucky vein deposits are generally agreed that they were formed from solution, and that the source of the minerals has been from the limestones of the region. It is not impossible that the source of some of the minerals at least may have been the magma from which the peridotite dikes were derived. Smith says:[18] "It is not probable, however, that the dikes themselves are the source of the fluorite, chiefly on account of the total volume of the fluorite veins as compared with that of the igneous dikes and the very small fluorine content of the peridotite."

Maryland

In Maryland barite occurs in Frederick and Carroll counties, and small occurrences have been reported in the Hagerstown Valley which are sim-

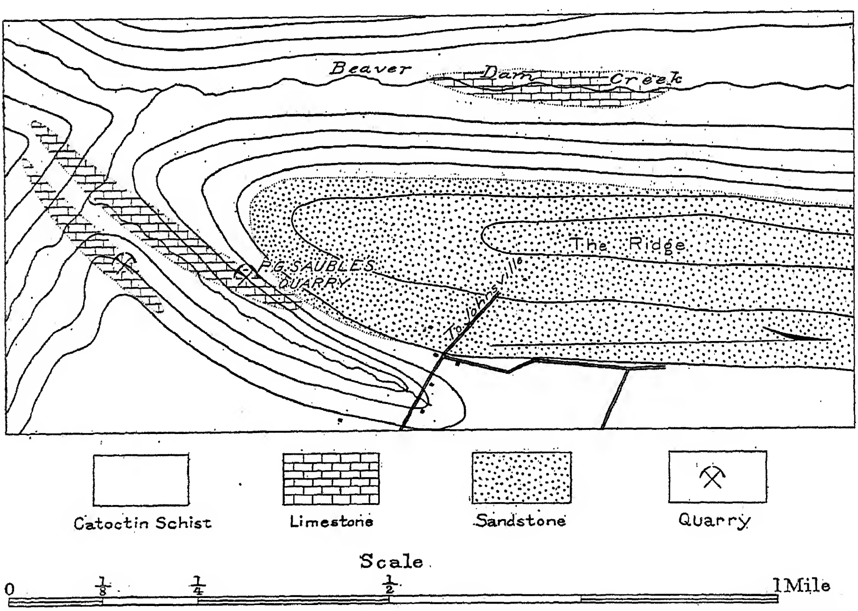

Fig. 5.—Map of Sauble Quarry and Vicinity, near Union Bridge, Maryland.

ilar to those found to the northeast in Pennsylvania. However, the deposits in Frederick and Carroll counties give the greater promise of eventually becoming of economic importance. The deposit here de-

[18] *Idem*, p. 151.

scribed may, broadly speaking, be considered as typical. Its association with the Piedmont limestones suggests that further search for barite in this particular physiographic province (Piedmont) would be worth while. Deposits similar to that on the Sauble property occur not only in the Piedmont limestones of Virginia but are also found at the same geologic horizon in North Carolina and South Carolina.

The Sauble property is located on a small feeder of Beaver Dam Creek in Frederick County and to the east of the large quartzite ridge extending in a northeast-and-southwest direction between Johnsville and Union Bridge. Structurally, the rocks of this ridge appear to be compressed into a syncline overturned to the westward. The quartzite is tentatively

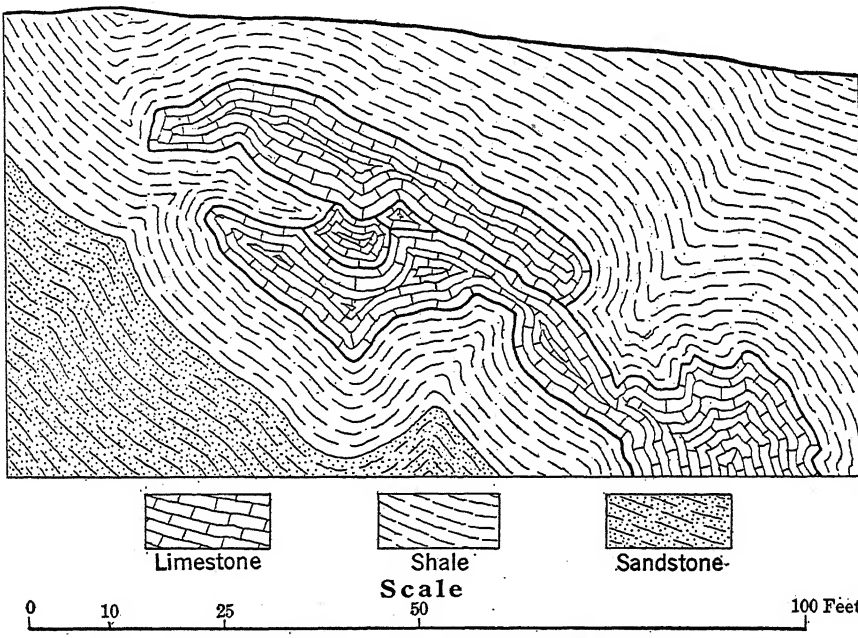

Fig. 6.—Section at Sauble Quarry, Maryland.

correlated with the Weverton formation and is underlain by the Loudon, which is composed here of limestone and shale. (Figs. 5 and 6.)

The barite on the Sauble property occurs in the shape of small bands, stringers, etc., in the Loudon limestone, which, as would be inferred from the structural relations referred to above, is greatly fractured and crushed, and in the overlying shale fracturing and crushing are even more emphasized. The quarry opened here has never been worked, however, for barite, but was operated with a view to obtaining limestone for calcination for local use, with no attention given to the barite. Judging from the lumps and small piles of barite lying about, it would seem that, to some extent at least, the mineral was sorted out and thrown aside.

Barite is also found at other localities in the Piedmont province of Maryland, and is characterized by occurring in crushed zones and along lines of dislocation which resulted from movements that profoundly compressed and more or less altered all the rocks of the region. These limestones were formerly much more widely distributed, but, in general, only the portions which have been compressed and preserved by reason of their synclinal structure now remain, and in most instances they have been greatly crushed and fissured, and recemented by calcite, and here and there by barite.

North Carolina[19]

North Carolina is one of the important barite-producing States in the Appalachian region. Deposits of barite occur in Gaston and Orange counties in the Piedmont region, and in Madison County in the western part or mountainous region of the State, but the production is confined to Gaston and Madison counties.

In Orange County, barite of good quality is found about 3.5 miles southeast of Hillsboro, near the Chapel Hill road. Some development work was carried on in 1901.

Barite is found at several localities in the vicinity of King's and Crowder's mountains, but the principal producing mine is located about 5 miles south of Bessemer City, Gaston County, near the north end of Crowder's Mountain. The barite occurs in quartz-sericite schist as veins ranging in thickness from 2½ to 6 ft. as exposed in the workings. The veins strike about N. 10° to 15° E., and dip at high angles, usually toward the west; in some places they are nearly vertical. Mining of barite has been carried on at a number of places in the vicinity of the Lawton mine, as shown in many old workings. Cyanite schist outcrops near the veins on the west side and lenses of magnesian limestone occur in the region, but so far as observed are not found in association with the barite.

The barite is granular, sometimes coarsely crystalline, and in places cleavage surfaces 2 in. across are noted. The associated minerals are quartz, galena, sphalerite, pyromorphite, and stains of iron and manganese oxides. According to Pratt, the veins are lenticular in shape, and probably represent the filling of fissures and crevices in the schists, which may have been caused by the faulting or tearing apart of the rock.

In Madison County in western North Carolina, the barite deposits occupy an area about 5 miles long extending a little north of east from Bluff on Spring Creek to and across French Broad River. Within this area barite has been worked chiefly in the vicinity of Marshall, Stack-

[19] Keith, Arthur: *Asheville Folio, North Carolina-Tennessee, No.* 116, *U. S. Geological Survey*, p. 9 (1904).

Pratt, J. H.: *Economic Paper No.* 6, *North Carolina Geological Survey*, pp. 62 to 65 (1902).

house, Sandy Bottom, and Hot Springs. According to Keith, the barite occurs as irregular veins in pre-Cambrian granites and to a less extent in

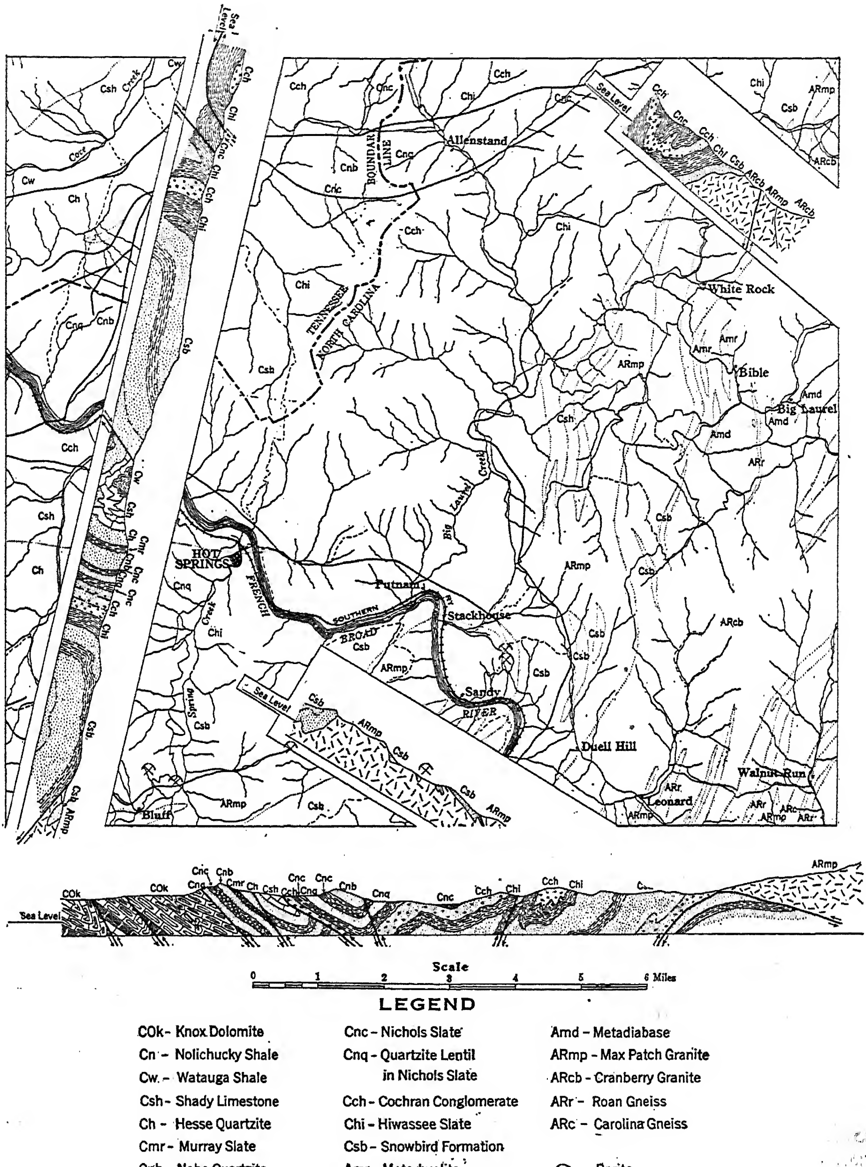

FIG. 7.—MAP AND SECTIONS OF THE BARITE AREA IN WESTERN NORTH CAROLINA. (*Asheville Folio, North Carolina-Tennessee, No. 116, U. S. Geological Survey.*)

Cambrian feldspathic quartzite. Watkins[20] reports that some of the deposits are associated with limestone. Pratt states that the veins vary

[20] Personal communication, October, 1914.

from 3 to 10 ft. in thickness and the barite is practically free from other minerals. At the Stackhouse mine Pratt reports that outcroppings of barite have been observed on the surface for a distance of 2,200 ft. The map, Fig. 7, gives a good general idea of the geology of the barite area in western North Carolina.

Keith gives the following description of the deposits:[21]

"The country rock was much broken and shattered before the formation of the barite, and the spaces were filled with the ore. From the arrangement and crystalline condition of the barite, it is evident that it was deposited from aqueous solutions. As to the derivation of the material in solution there is no evidence; probably it came from the great mass of granite within which it is now found. All of these barite areas occur in a belt near and parallel to one of the principal thrust faults of the region. The rocks were greatly disturbed during the production of the original fault plane, and later, to a less extent, during its secondary deformation. It appears most likely that the crushing of the quartzites seen in the barite deposits was associated with this later deformation. In the old workings on Spring Creek, near the level of the creek, the barite-bearing quartzites are very close to the thrust fault and there seems to be a distinct connection between the fault and the barite. The barite occurs in large crystals and crystalline masses which have not been deformed, as were the minerals of the inclosing rocks. From this it is clear that the barite was deposited after the period of faulting and folding, and although the shattering of the rock may have been due to the folding, the barite must have been introduced later. The crevices in the shattered granite afforded comparatively easy access to the solutions that carried the barite, but no direct connection in origin can be traced between the faults and the barite deposits.

"At present only the deposits on Spring Creek are being mined. A considerable amount of barite was taken from the veins a mile southeast of Stackhouse, but the deposits on Doe Branch were never developed to any extent. Southeast of Stackhouse small open cuts were made and short shafts and tunnels were driven. These were nearly on the crest of a ridge, and penetrated for some distance the more or less weathered granites. On Spring Creek the mining has been done chiefly through open cuts. These are now of considerable size and much material has been removed. The openings lie on ridges which are about 500 feet above water level and which slope steeply toward Spring Creek. The surroundings are thus favorable for extensive operations, and the quantity of barite appears to be large."

Pennsylvania[22]

Barite is not mined on a commercial scale in Pennsylvania at present, although small shipments have been made from the Cumberland Valley in the southern part of the State in the vicinity of Waynesboro, Franklin County. Barite is also reported from west of New Hope, Butler County; at Phœnixville mines, and in small quantity in old Jug Hollow mine, Schuylkill township, Chester County; formerly mined at Fort Littleton, Fulton County; and at the Perkiomen copper mines, Montgomery County.

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[21] Keith, Arthur: *Asheville Folio, North Carolina-Tennessee, No.* 116, *U. S. Geological Survey*, p. 9 (1904).

[22] Stose, G. W.: *Bulletin No.* 225, *U. S. Geological Survey*, pp. 515, 516 (1904).

Sanford, S., and Stone, R. W.: Useful Minerals of the United States, *Bulletin No.* 585, *U. S. Geological Survey*, p. 158 (1914).

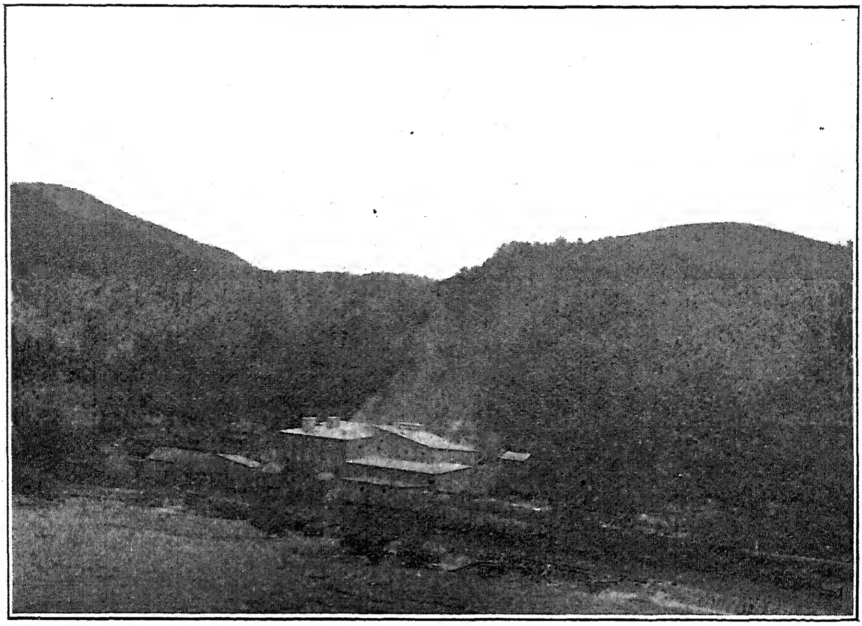

FIG. 8.—BARITE MILL OF THE CAROLINA BARYTES CO., STACKHOUSE, N. C.
(J. H. WATKINS, PHOTO.)

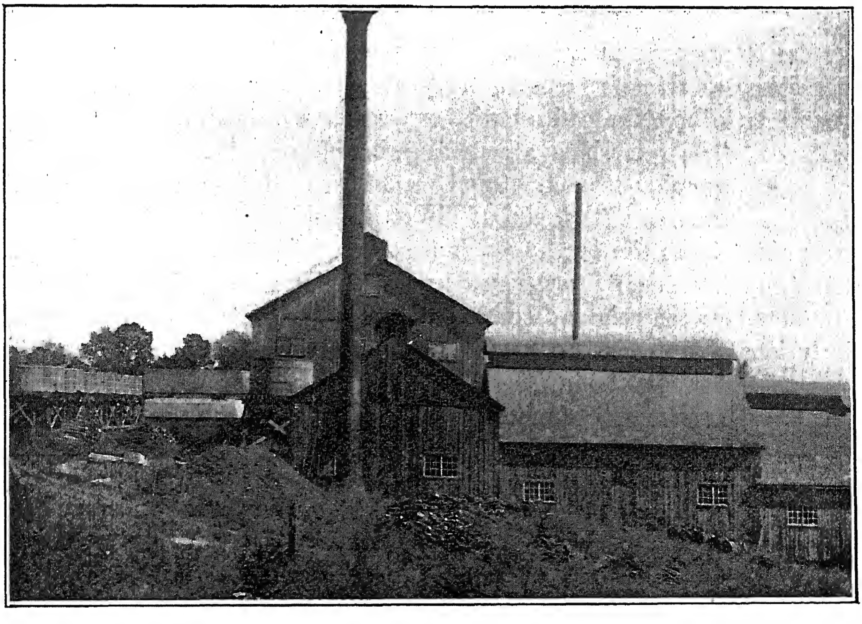

FIG. 9.—BARITE MILL OF THE CHEROKEE CHEMICAL CO., KING'S CREEK, S. C.
(J. H. WATKINS, PHOTO.)

In the vicinity of Waynesboro the barite is associated with Cambrian limestones and their residual clays. Openings have been made to the north, east, and northeast of Waynesboro both in the residual clays and in the limestone. The deposits are found in the folded and brecciated portions of the limestone, and the barite occurs in the brecciated rock as a vein filling, but such deposits are not considered profitable. Stose says of the barite in the two openings made in bedrock (limestone) visited by him: "Here the barite is chiefly massive, banded, sub-crystalline or granular, and milky, resembling chert, but is in part clear and crystalline." In the residual deposits the lumps or masses of barite average 6 to 8 in. in diameter, are rough in outline with evidence of dissolved limestone fragments, and are considerably weathered.

South Carolina[23]

The known commercial deposits of barite in South Carolina are confined to Cherokee County on the North Carolina line. The production has been small and irregular, not, however, on account of lack of quantity and quality of the mineral. The deposits are located in the vicinity of Kings Creek, a station on the Blacksburg-Yorkville branch of the Southern Railway, and are about 13 miles southwest of the Lawton mine in Gaston County, North Carolina (see p. 363).

The barite occurs in quartz-sericite schist as lenticular veins. Thus far limestone has not been observed in association with the barite deposits. The development work done comprises open cuts and pits, which expose in places as many as three parallel veins ranging from 1 to 3 ft. in thickness. Watkins reports the greatest thickness observed was more than 6 ft. of massive white barite exposed in an open cut north of the railroad. The veins have a general strike of about N. 25° E., and dip about 40° to 45° southeast. On the property of the Cherokee Chemical Co., the veins east of the railroad dip to the southeast while those on the west side dip to the northwest, indicating erosion of an anticlinal fold. They conform roughly to the structure of the inclosing schists, and sharp contacts between vein and country rock are usually shown, though small inclusions of the schist are incorporated in the barite in places. Discordance of strike is noted in places, as the veins cut the schists at a slight angle. Branching veinlets are given off from the main veins which fill fractures that cross the structure of the schists. (See Fig. 10.) The field evidence indicates that the veins have had a similar genesis to those farther northeast in Gaston County, North Carolina (see p. 363).

The barite is of good quality, massive granular in structure, and varies

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[23] Sloan, Earle: Catalogue of the Mineral Localities of South Carolina, *Bulletin No. 2, ser. iv, South Carolina Geological Survey*, pp. 125, 126 (1908).

Unpublished data funished by Joel H. Watkins, geologist for the Southern Railway Co., Washington, D. C.

from pure white to pink in color, the white predominating. The pink is mixed with the white barite in milling, since there is apparently no difference in quality of the two differently colored materials. Some parts of the veins are very pure, being singularly free from associated minerals, but other parts contain a little galena and less fine-grained quartz. Sloan states that occasional inclusions of pyrite cause local areas of stain. The barite is generally of good bleaching quality.

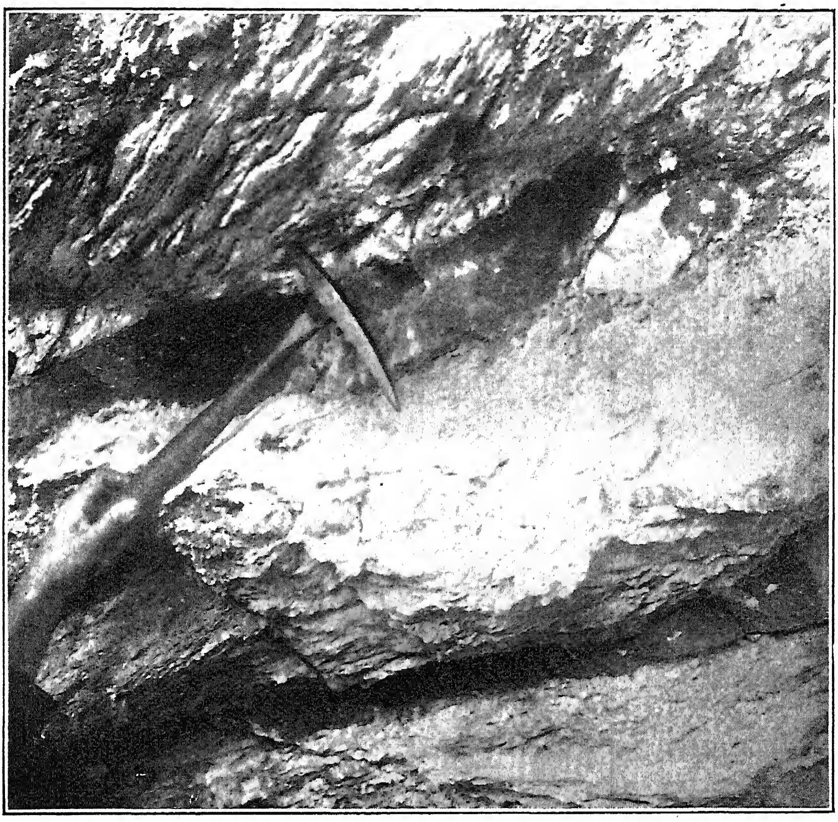


Fig. 10.—Small Veinlet of Barite from a 3-ft. Vein, Cutting Across Structure of Schist. Cherokee Chemical Co., Kings Creek, S. C. (J. H. Watkins, photo.)

The mill of the Chrokee Chemical Co. is located directly alongside the railroad track and the deposits have been opened at a number of points both to the north and south of the mill, with most of them lying within a radius of half a mile therefrom. The following analysis by Ricketts & Banks of a representative sample of the white barite collected by Banks from the property of the Cherokee Chemical Co., located at Kings Creek, indicates its composition: $BaSO_4$, 93.59; Fe_2O_4, 0.32; CaO, 0.02 per cent.; MgO, trace; SiO_2, 4.76 per cent.

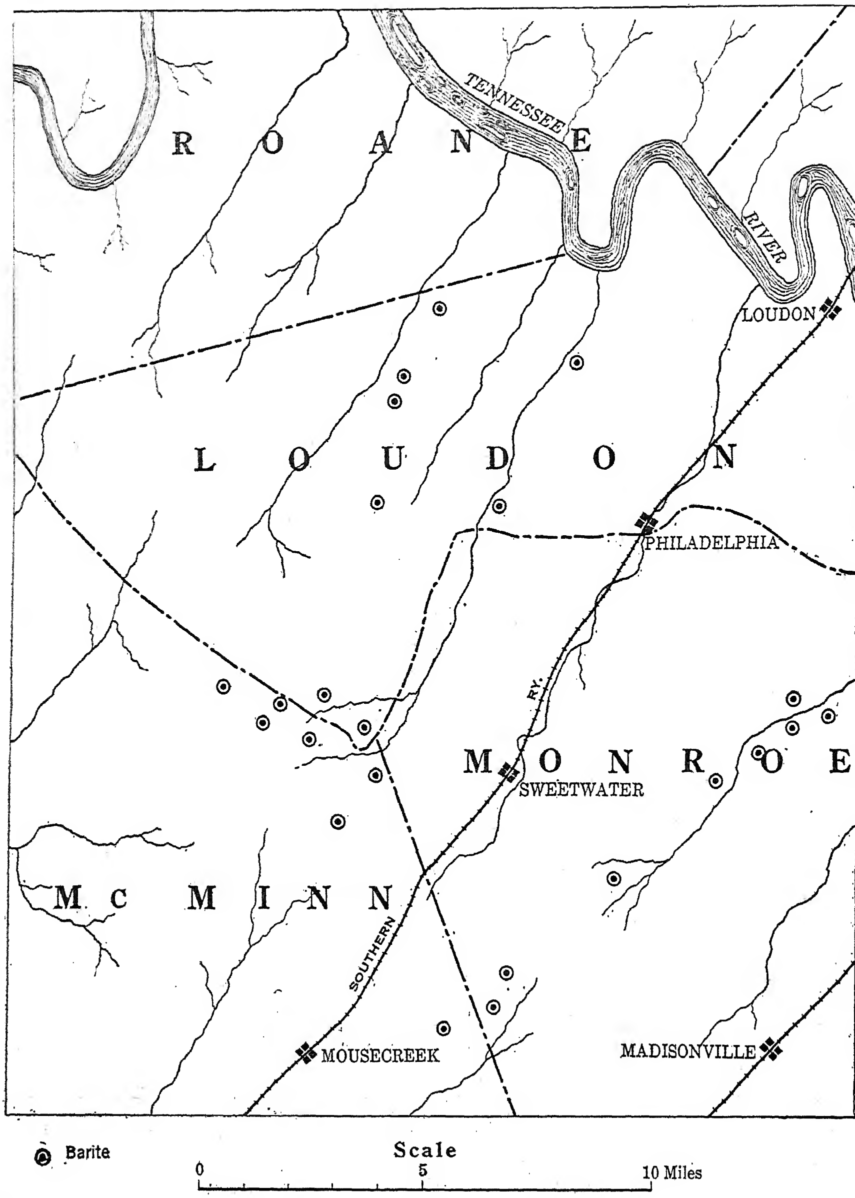


FIG. 11.—MAP OF THE SWEETWATER DISTRICT, TENNESSEE, SHOWING LOCATION OF BARITE WORKINGS. (ADAPTED FROM THE TENNESSEE GEOLOGICAL SURVEY.)

Tennessee[24]

In Tennessee barite is found in many localities, but the more important deposits occur in the eastern part of the State and are grouped into two districts: (1) The French Broad district, which includes parts of Cocke and Sevier counties on the North Carolina line, and (2) the Sweetwater district, centered about the town of Sweetwater and including parts of Loudon, McMinn, and Monroe counties. Barite has been reported

FIG. 12.—1,000 CARLOADS OF BARYTES, SWEETWATER, TENN. (G. F. GREENE, PHOTO.)

mined also in Bradley, Jefferson, Smith, and Washington counties; and it occurs as a gangue mineral in a lead mine near Haysborough, Davidson County, and as veins in dolomite 12 miles from Greeneville, Green County.

[24] Fay, A. H.: Barytes in Tennessee, *Engineering and Mining Journal*, vol. lxxxvii, No. 2, p. 137 (Jan. 9, 1909).

Grasty, J. S.: The Barite Deposits of Tennessee, *The Tradesman*, vol. lxix, pp. 34 to 38 (1913).

Henegar, H. B.: Barite Deposits in the Sweetwater, Tennessee, District, *The Resources of Tennessee, State Geological Survey*, vol. ii, pp. 424 to 429 (1912).

Herzig, C. S.: Tennessee Barytes, *The Mineral Industry*, vol. x, p. 58 (1901).

Judd, E. K.: The Barytes Industry of the South, *Engineering and Mining Journal*, vol. lxxxiii, No. 16, pp. 751 to 753 (Apr. 20, 1907).

Watson, Thomas L.: Fluorite and Barite in Tennessee, *Trans.*, xxxvii, 890 (1906).

Weller, C. A.: Barytes Mines of the Commercial Mining and Milling Company (Tennessee), *Engineering and Mining Journal*, vol. lxxxiii, No. 18, p. 851 (May 4, 1907).

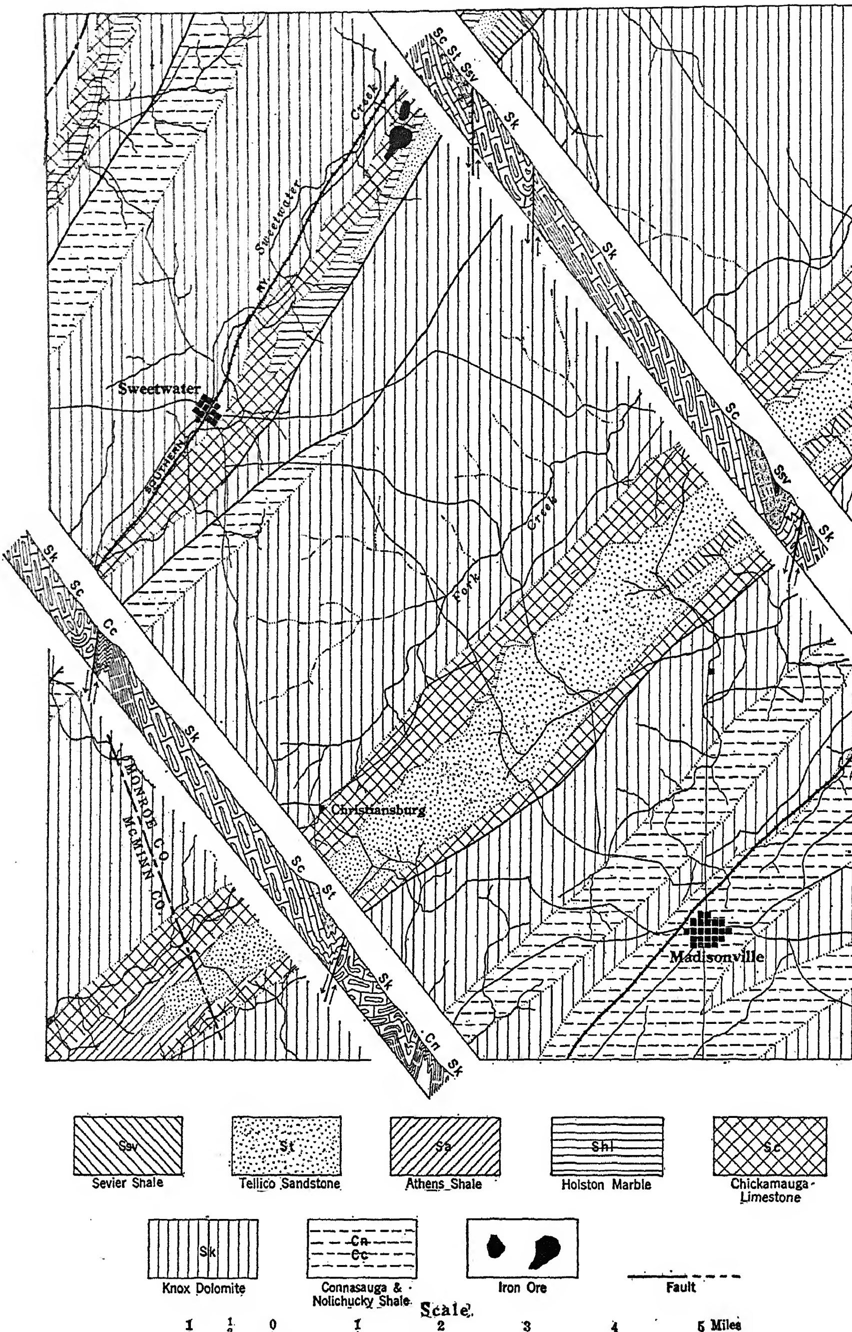


FIG. 13.—GEOLOGICAL MAP AND SECTIONS OF SWEETWATER DISTRICT, TENNESSEE. (*London Folio, Tennessee, No. 25, U. S. Geological Survey.*

The French Broad district includes parts of Cocke and Sevier counties, located on the North Carolina line in the mountainous region of the State, south and a little east of Knoxville. The barite occurs in veins from 1 to 6 ft. in thickness, which closely resemble those of Madison County, North Carolina, but owing to lack of transportation the deposits have only been worked to a limited extent. Thus far, the principal developments have been near Wolf Creek in Cocke County. The deposits are reported to contain ample reserves of the mineral, and they carry practically none of the impurities associated with the residual deposits of barite in the Sweetwater district.

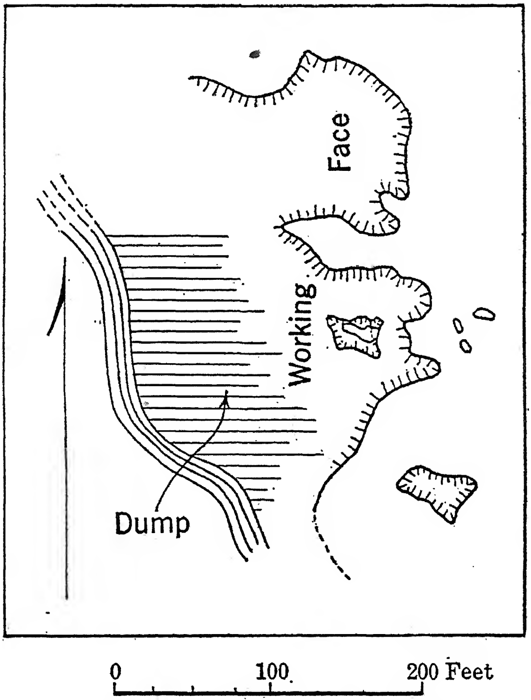
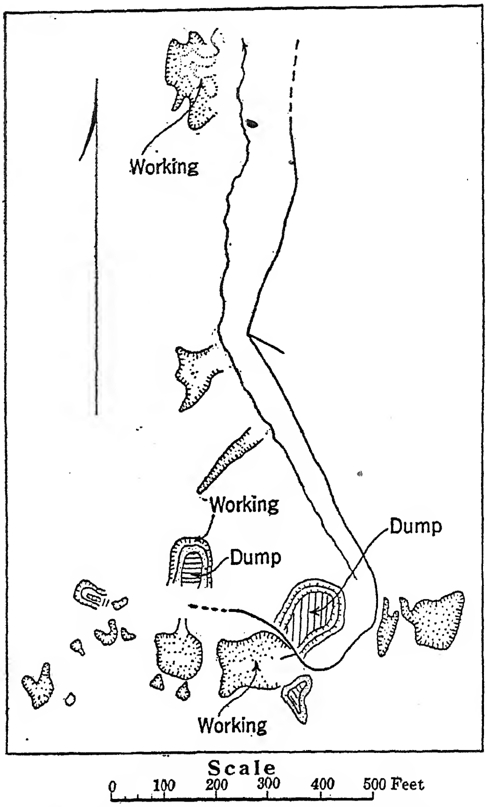


Fig. 14.—Sketch Map of Barite Workings on Terry Property.

The Sweetwater district is the largest and at present the only important producer of barite in Tennessee. It includes parts of Loudon, McMinn, and Monroe counties, centered within a radius of 20 miles about the town of Sweetwater on the Knoxville division of the Southern Railway, 42 miles southwest of Knoxville. (See Fig. 11.) There has been extensive development and a considerable production of barite in the past. The principal shipping point is Sweetwater, where is located the mill of Gilman & Son for the treatment of the mineral, but other shipping points located on the same line of railway include Philadelphia, Reagan, and Niota. Most of the barite is graded and shipped to the Eastern markets.

(See Fig. 12.) Several companies operate in the district, but the mineral is handled almost entirely by farmers who work between crops.

The barite deposits of the Sweetwater district are of the residual type, occurring as nodules and boulders of varying sizes and shapes in the clays derived from the weathering of folded and faulted rocks which range in age from Cambrian to Ordovician. (See map, Fig. 13.) The principal workable deposits, however, are found in the residual clays derived from the Knox dolomite. In places the mantle of residual clay, in which



[Fig. 15.—Method of Mining Barite from Clay in Tennessee, Ray Property.

are assembled the nodules and boulders of barite, is 60 ft. and more in thickness, but usually averages from 10 to 15 ft. Only in one or two workings have the excavations extended down to the hard rock limestone.

A large percentage of the barite mined in the district is coated with limonite, which on account of the slight difference in specific gravity of the two minerals is difficult to remove even by jigging. Freeing of the barite from the iron oxide before grinding is necessary, as it is very difficult to bleach the mineral thoroughly afterward. Besides iron oxide the barite is intimately associated with chert.

An analysis of a sample of barite taken from the Ray property gave: $BaSO_4$, 97.56; Fe_2O_3, 0.42; SiO_2, 0.41 per cent. A less favorable sample from the Terry property gave on analysis: $BaSO_4$, 94.12; Fe_2O_3, 0.70; SiO_2, 2.45 per cent.

Mining of barite in the Sweetwater district is carried on entirely by open-pit work with pick, shovel, wheelbarrow, and wagons. Figs. 14 and 15 serve to show the general character of mining in winning barite from residual clays in the district. In some cases the clay encountered is indurated, and because of its toughness it is necessary to break it down by an explosive. This, however, is the exception, although an explosive might be used at times to advantage in even the less-hardened clays.

Virginia[25]

In Virginia, barite has been mined since 1845. In the Piedmont region the mineral has been mined in Bedford, Campbell, Louisa, Pittsylvania, and Prince William counties; and in the Valley region in Montgomery, Russell, Smyth, and Tazewell counties. The barite deposits occur in three unlike areas: (1) In the red sandstone-shale series of the Triassic, (2) in the old crystalline metamorphic pre-Cambrian and Cambrian rocks, especially in some of the crystalline limestone areas, and (3) in the Valley region of folded and faulted Cambrian and Ordovician limestones. Areas (1) and (2) compose the Piedmont province, which extends eastward from the Blue Ridge to the fall-line or the western margin of the Coastal Plain sediments. The map, Fig. 16, shows the distribution of worked barite areas in Virginia.

Thus far the Prince William County deposit near Catlett, in the northern part of the State, is the only one in the Triassic rocks that has been worked to any extent. It has been idle since 1903. The barite occurs as a filling or cement of crushed and fractured impure limestones and red shales. The widest of the barite-filled fractures is 4 to 8 ft. The barite is of good white quality, finely to coarsely crystalline massive, and is largely free from most of the common impurities. The mineral was mined by shafts and open cuts, the greatest depth reached being 108 ft.

The principally worked barite deposits of Bedford, Campbell, and Pittsylvania counties, in the south-central part of the State, are centered about Evington, Campbell County, and Toshes and Sandy Level, Pittsylvania County, as shown on map, Fig. 17. The area is one of greatly altered crystalline rocks formed from old sedimentary and igneous rocks.

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[25] Watson, Thomas L.: Geology of the Virginia Barite-Deposits, *Trans.*, xxxviii, pp. 710 to 733 (1907).

———: The Geology of the Virginia Barite Deposits, *The Tradesman*, Mar. 20, 1913, pp. 38 to 41, Mar. 27, 1913, pp. 31 to 33.

Campbell, M. R.: *Bristol, Virginia-Tennessee, Folio, No.* 59, *U. S. Geological Survey* (1899).

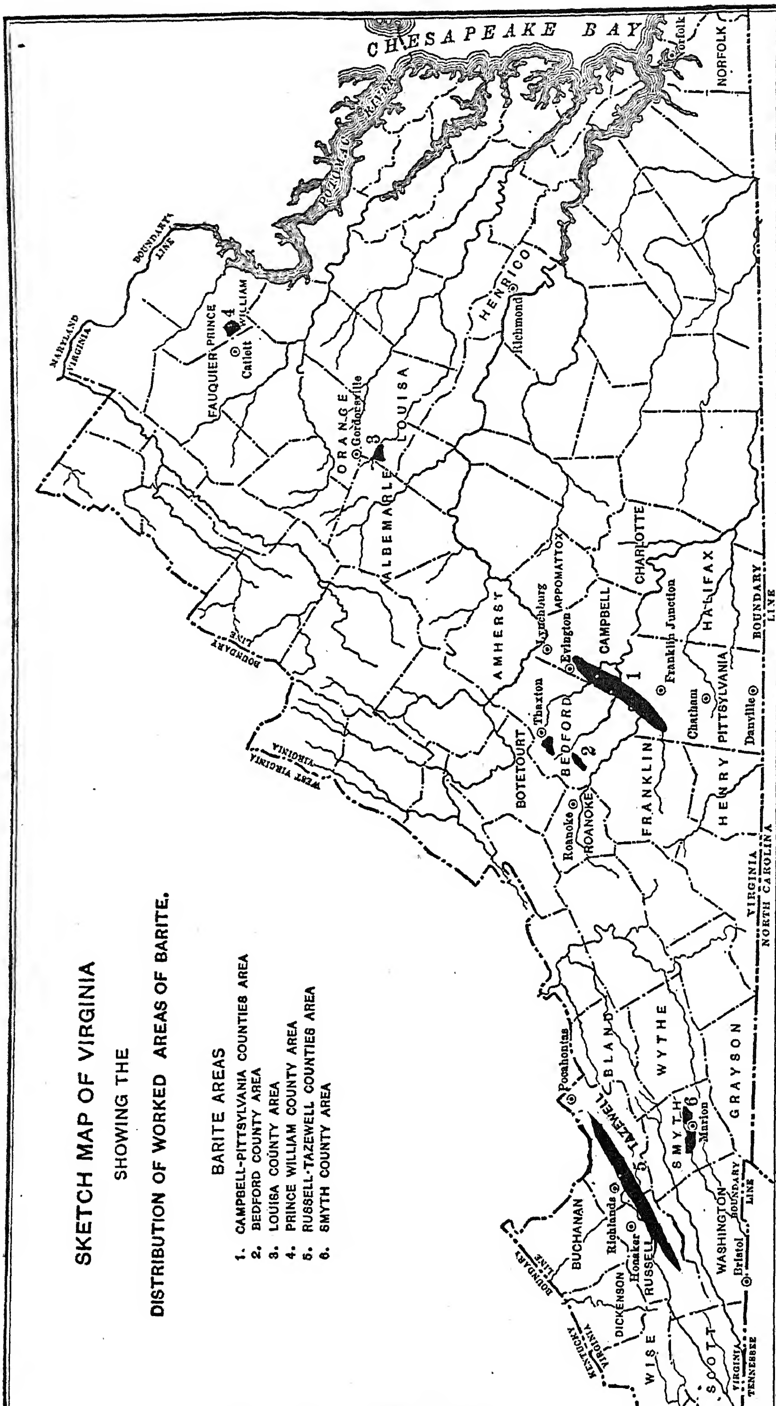

FIG. 16.—SKETCH MAP OF VIRGINIA, SHOWING BARITE AREAS.

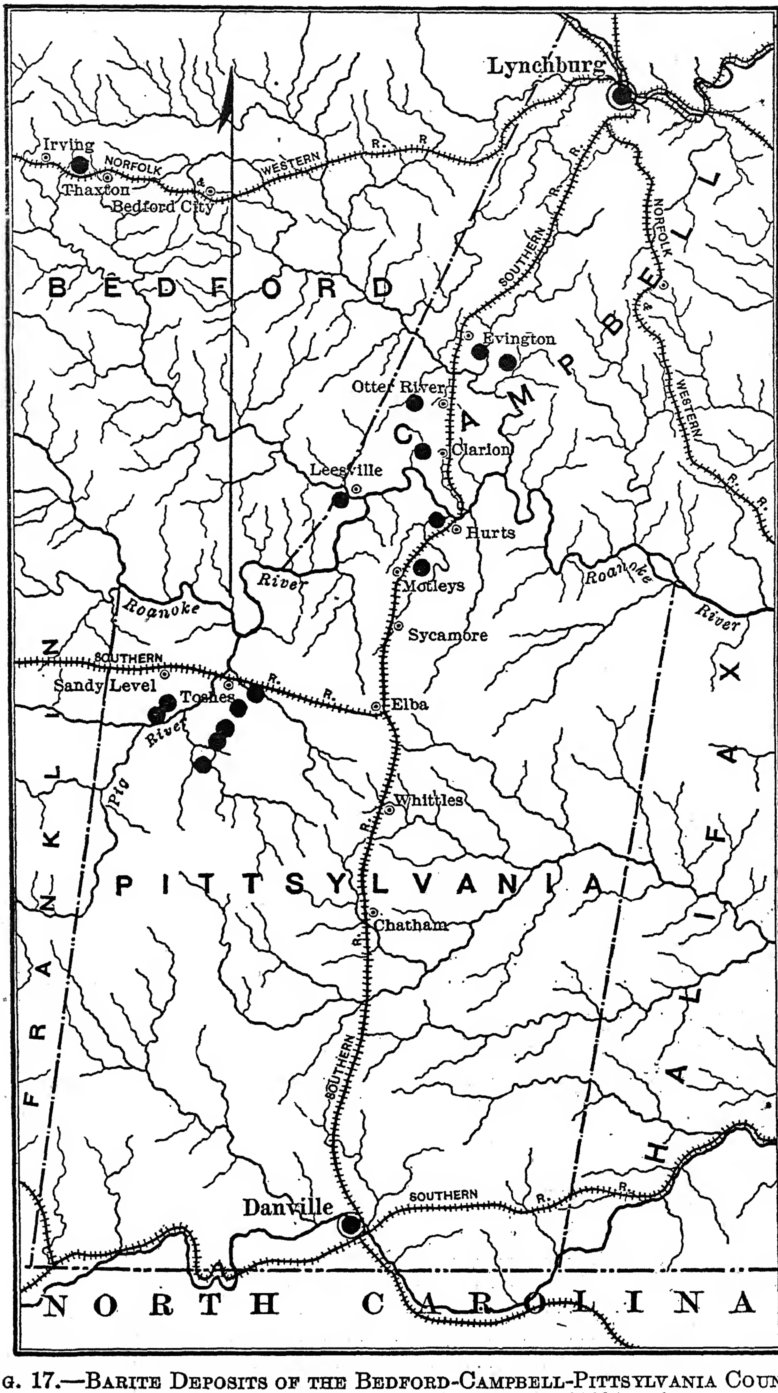

FIG. 17.—BARITE DEPOSITS OF THE BEDFORD-CAMPBELL-PITTSYLVANIA COUNTIES AREA, VIRGINIA. Scale, 1 in. = 7.5 m., approximately.

The barite occurs as irregular lense-like bodies or pockets replacing crystalline limestone associated with micaceous and quartzose schists, and as nodules and lumps in the residual clays, which are limonitic or manganiferous and have been used in the manufacture of natural mineral pigments. Besides calcite and silicate minerals of the inclosing rocks, the associated minerals are galena, pyrite, and less sphalerite and chalcopyrite, in the unaltered deposits; and manganese oxide and iron oxide in the residual deposits. These impurities are often developed only sparingly and are not noticeable in the best grades of the barite. Fig. 18, an ideal section

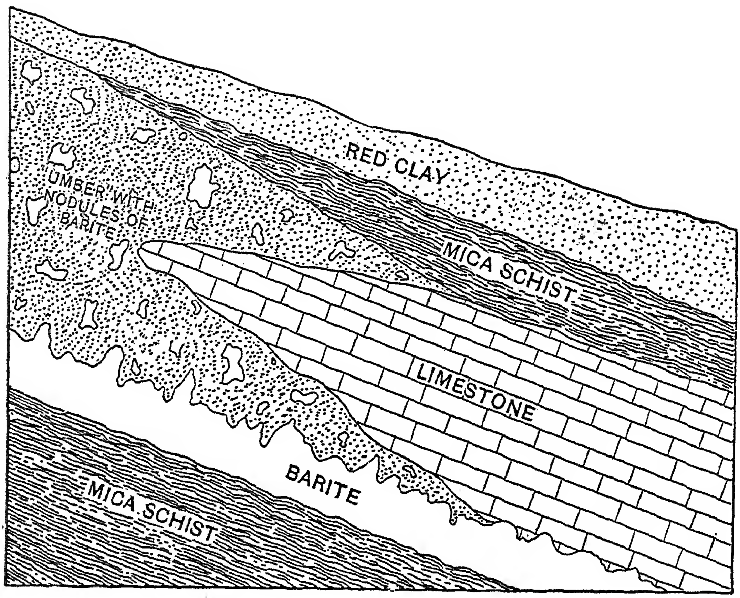

FIG. 18.—IDEAL SECTION IN BENNETT BARITE MINE, PITTSYLVANIA COUNTY, VIRGINIA.

in the Bennett mine, Pittsylvania County, illustrates the mode of occurrence of the mineral in Campbell and Pittsylvania counties.

A sample of the black manganiferous clay collected from the Bennett mine, Pittsylvania County, by the senior writer and analyzed by W. B. Ellett, gave:

	Per Cent.
Insoluble residue	14.20
Alumina	4.96
Ferric oxide	32.40
Manganous oxide	19.49
Lime	2.06
Magnesia	trace
Barium oxide	trace
Copper	trace

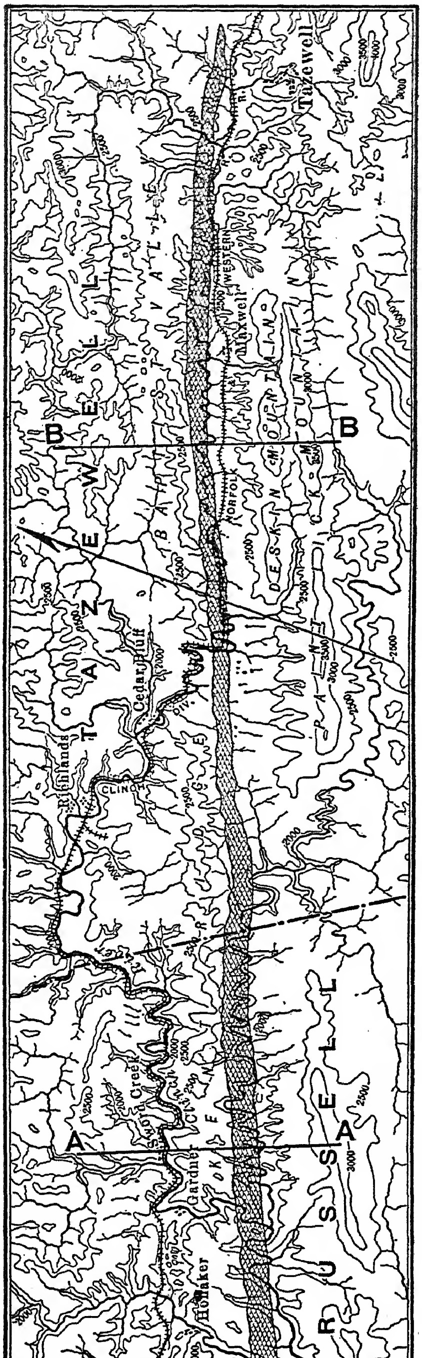

FIG. 19.—MAP OF A PART OF TAZEWELL AND RUSSELL COUNTIES, VIRGINIA, SHOWING POSITION OF GREATER PART OF BARITE ZONE. CHECKERED AREA THROUGH CENTER OF MAP IS BARITE. ADAPTED FROM THE ECONOMIC GEOLOGY SHEET OF THE *Tazewell Folio, U. S. Geological Survey.* Contour interval, 500 ft. Scale, 0.25 in. = 1 mile, approximately.

The composition of the associated crystalline limestones is shown in the analyses below, made by W. B. Ellett on samples collected by the senior writer.

	1 Per Cent.	2 Per Cent.	3 Per Cent.
Insoluble matter	1.66	0.87	1.10
Alumina, Iron oxide	0.24	0.30	0.96
Barium sulphate	0.62	0.65	1.62
Calcium carbonate	89.36	93.33	91.07
Magnesium carbonate	6.61	2.82	3.73
Copper sulphide	trace	trace	0.36

1 and 2. White crystalline limestone from the Hewitt mine, Campbell County.
3. White and pink crystalline limestone from the Ramsay mine, Pittsylvania County.

The barite is won chiefly by shafts and drifts, and open cuts and pits. The greatest depth reached is 160 ft., at the Hewitt mine in Campbell County.

Barite is found in association with similar rocks in other counties of the Piedmont province, but has not been mined to any considerable extent. In the western part of Bedford County, about 3 miles northwest of Thaxton, barite has been mined from a fracture-filled deposit in a coarse

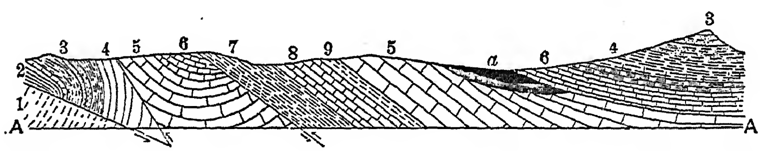

Fig. 20.—Structure Section Along Line A–A of Fig. 19, Showing Structural Relations of the Barite and Rocks. *a* is Barite. Adapted from *Tazewell Folio, U. S. Geological Survey.*

granite. The mineral is crystalline, white to blue-gray in color, and in places contains much disseminated galena in small grains and occasional sphalerite. In Louisa County coarsely crystallized barite has been mined 3 miles south of east from Lindsay. It occurs in veins in schists and is remarkably free from impurities other than the usual discoloration from red iron oxide and some quartz. Several promising deposits of barite in the vicinity of Bealeton, Culpeper County, have been opened but no shipments have been made.

In the Valley region of southwest Virginia, in Tazewell, Wythe, Russell, and Smyth counties, barite occurs in the Cambrian and Ordovician limestones and their residual clays. The mineral occurs in the limestone as replacements and veins associated with sphalerite, galena, pyrite, and some fluorite. In Tazewell and Russell counties the common associates are limonite, chert, and calcite, with some siderite, and occasional fluorite.

Barite has been extensively mined near Marion, Smyth County, and extensive deposits are found in Russell and Tazewell counties, chiefly along the southern slope of Kent Ridge and its prolongation northeast and southwest, along the valley of Clinch River, extending from near North Tazewell to near Lebanon, a distance of more than 30 miles. The mineral has been mined at several points along this belt, the principal mines being

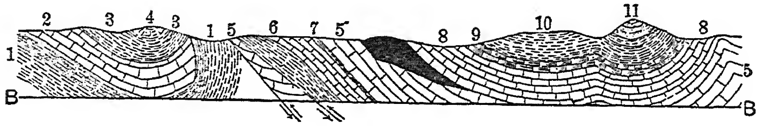

Fig. 21.—Structure Section Along Line B–B of Fig. 19, Showing Structural Relations of the Barite and Rocks. Black is Barite. Adapted from *Tazewell Folio, U. S. Geological Survey.*

near North Tazewell, 3 miles south of Richlands, 3 miles from Honaker on the Clinch River, and at the southwestern end of the belt near Lebanon. The barite is found in the upper part of the Knox dolomite. Figs. 20, and 21 show the structural relations of the Knox dolomite and the adjacent rocks on the northwest and southeast, near Sword Creek and Richlands, in Tazewell County.

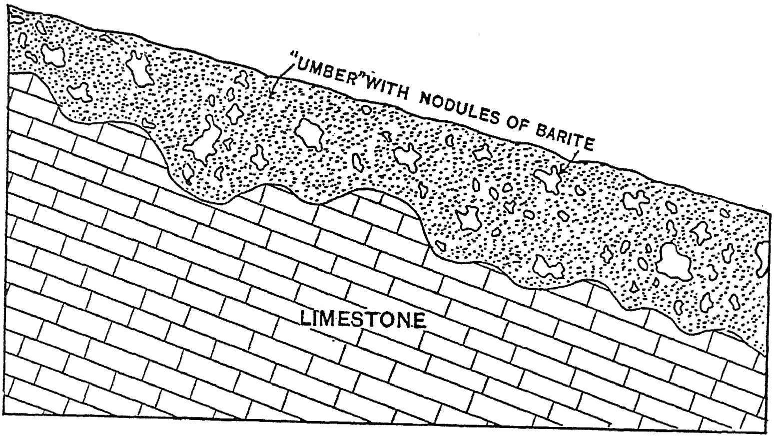

Fig. 22.—Principal Mode of Occurrence of Barite in Russell and Tazewell Counties, Virginia.

The barite occurs as lumps of varying shapes and sizes assembled in the residual clays of the limestone, and in pocket form and vein-like bodies filling spaces in the limestone, and in part as replacements of the limestone. Fig. 22 illustrates the residual mode of occurrence of the barite. The mineral is crystalline, of good white quality, and in most places is practically free from impurities. The greatest depth reached in mining is 103 ft., at the mines of the Pittsburgh Baryta & Milling Corporation on

the northeast end of the belt. Most of the mining done has been for the barite nodules in the residual clays, won from shallow open pits and cuts. Some hard-rock mining in the limestone has been done in places.

Mining Methods

Mining of barite in the Appalachian States is chiefly surface work, with no really deep mining in any part of the region. Thus far the mineral has been won chiefly by open cuts, shallow shafts, and pits, which rarely exceed 40 ft. in depth, usually much less. This method of winning the mineral is especially applicable to the occurrence of barite in loose nodular masses of different sizes and shapes irregularly assembled in the residual clays.

In some districts, especially the Piedmont region of Virginia and central Kentucky, shafts ranging from 50 to 400 ft. in depth have been sunk. In Piedmont, Virginia, the barite is won by vertical timbered shafts and drifts which follow the mineral bodies, and the greatest depth yet reached in barite mining is less than 200 ft. In the central Kentucky region a depth of about 400 ft. has been reached but the shafts have been chiefly for prospecting. Tunneling has been practiced to some extent in this region in the high cliffs bordering the Kentucky River, where the veins are exposed for a vertical distance of 300 to 500 ft.

Blasting is necessary for breaking down the barite in the vein and limestone replacement occurrences of the mineral.

Development and Trade Conditions

The majority of the deposits of barite are so scattered and many of them so small that the problems connected with their operation are those of handling the mineral and getting it to market rather than of mining. However, there is room for much improvement in the methods of mining barite, as is evidenced by the large percentage of loss in the working of deposits that occur in residual clays. This loss, which in some cases amounts to as much as 30 per cent., is from the small lumps and grains recoverable by washing and jigging but too small for cutting and sorting by hand. A large number of the deposits are located at a distance from railway transportation, as well as being far removed from mills, and while this is a disadvantage that obtains in many instances, still many deposits occur which would be worked were not the freight rate the main deterrent. On the other hand, the problem that confronts the manufacturer at times is the difficulty of accumulating sufficient barite to keep the mill running continuously. The reason why the raw material is not always available is due to the fact that the barite producer lacks, because of decreased profits, the same incentive that actuates the manufacturer. Moreover, the foreign producer has found that he can deliver his product to the larger

city markets, like New York, Philadelphia, Baltimore, and others, cheaper than the domestic producer can deliver by rail.

Where barite occurs in residual clays the best way to handle it with a sufficient quantity of ore in sight would be with a steam shovel and suitable washers, but since few, if any, deposits are large enough to warrant the installation of such a plant and water is not everywhere plentiful, the slower methods of mining, cobbing, and rocking must be employed. However, by employing energetic methods resulting in cheaper mining and the winning of a large percentage of the mineral that goes now to the dump heap in shape of small lumps and grains, the barite operators could increase considerably their present margin of profit, and thus operate profitably a large number of the deposits now idle.

Methods of Preparation

The preliminary treatment of barite for the market will vary according to the character of the mineral. In some localities the barite contains little or no impurities, in others the amount and character of the impurities vary greatly. The minerals associated with barite vary locally, but, broadly speaking, they vary in accordance with the mode of occurrence, whether residuary, or as veins and replacements. The more commonly occurring minerals are iron oxide, manganese oxide, limestone including calcite, and silica in the form of both quartz and chert, in the residual deposits; and the metallic sulphides, especially galena and sphalerite, fluorite, and calcite, in the vein and replacement deposits.

For the removal of impurities from the merchantable barite the principal steps in the preparation for market include hand cobbing, sorting or grading, washing, and crushing. For ground barite, bleaching, drying, and grinding are necessary. Such associated minerals as galena, quartz, calcite, and limonite, can be removed largely by hand cobbing during mining, but if these are inclosed in the barite, crushing and washing may often serve to cleanse it. Jigging is often found to be advantageous, especially if the iron oxide is in the form of scales or crusts, and it may be successfully employed for removing impurities of light weight, such as chert, limestone, etc. After the mineral has been hand cobbed at the mine into the several grades, it is shipped to the mills for further cleansing by crushing and washing to remove as much of the impurities as possible.

Much of the barite mined in the Appalachian States is won by open-cut work in residual clays derived from limestones; hence, the barite nodules are usually more or less stained with iron oxide and coated with clay. A method often practiced for freeing the mineral of these, especially the clay, is to assemble the nodules and lumps of the mineral in piles exposed to the weather, after which it is better prepared for hand cobbing and is given such additional cleansing as necessary prior to shipping to

the mills. In some cases washing the mineral in log washers is practiced, but this is not always feasible, since an ample water supply may not be available, and also loss of the fine material is likely to result from this treatment.

In some localities in the Appalachian States the washers installed have been crude and the management poor, hence poor results have been obtained. When feasible, however, log washing and jigging would yield a product vastly superior to that produced by sun-drying methods. Washing will not entirely cleanse the ore in all cases, especially when the nodules and lumps are coated with a heavy crust of iron oxide, as is often the case with the residual deposits, but it will appreciably reduce the iron content and correspondingly increase the price demanded.

Milling

Crude barite is treated by a number of different processes preparatory to a considerable range of uses. The equipment of the mill will depend upon whether the object be to prepare and market a ground and purified product, or to carry the treatment further by employing the comminuted material in the manufacture of compounds into which barium enters as an essential ingredient.

The barite as mined may be treated by either the wet or the dry process, the object in both cases being to reduce it to an impalpable powder and eliminate impurities, the two most difficult and deleterious being calcium carbonate and iron oxide. The former reacts injuriously with the acids of paints and the latter is objectionable because of its color.

In case the wet process is employed, suitable crushing, pulverizing, and jigging machinery is installed and the barite is kept in contact with water from the beginning to the completion of the treatment. When it is sought, however, to prepare the ground and purified product by the so-called dry method, the dry material is first passed through crushers and then cleansed as far as possible by water in log washers. The log washer, however, only partly eliminates such impurities as galena, sphalerite, iron oxides, calcite, fluorite, etc., hence it becomes necessary to bleach the material in an acid solution.

An outline of the complete process as followed by a thoroughly equipped mill has been described by Burchard as follows:[26]

"The crude material is ground in slip mills having granite grinders and granite bases. Water is fed into these mills and the ground material is floated over the top of the tanks, after which it is pumped into funnel-shaped separators. The contents of the separators are agitated by flowing water and the coarser rejected material is drawn off at the bottom of the funnel and returns to the slip mills, while the finer material floats off at the top of the separators. This material next descends to

[26] Burchard, E. F.: *Mineral Resources, U. S. Geological Survey*, pt. ii, p. 687 (1907).

settling tanks, and after forming a sludge is drawn off into bleaching tanks. The bleaching tanks are built of concrete lined with refractory tile. Bleaching is accomplished by the addition of measured weights of sulphuric acid to the sludge and the agitation of the mass to secure thorough mixture. The acid reacts on the iron oxide and lime present, forming ferrous sulphate and calcium sulphate. The iron salt being soluble and the calcium salt partially soluble, besides having a lower specific gravity than the pure barytes, these substances, together with the excess of sulphuric acid, are removable in the further washing process to which the material is subjected. For this next washing the material is pumped into washers which employ the float-separation process. Next, the bleached barytes passes to settling tanks, after which it is dried by being spread thinly on the surface of a rotating hot drum. From the hot drum the dried material falls or is brushed off and carried to Williams mills, where it is pulverized, screened, and finally sacked by machine. The essential difference between this process and the others mentioned above is in the fact that the material is first reduced to a fine condition before bleaching, thereby bringing the sulphuric acid intimately into contact with all portions of the barytes."

Bleaching

Nearly all crude barite requires bleaching before the requisite degree of whiteness can be secured for the finished product. In order to free barite from iron stain, bleaching with sulphuric acid is necessary. The mineral is first crushed and washed, when it is ready for bleaching. The degree of fineness to which the mineral is crushed will depend upon the amount and character of iron present. In the Appalachian States, the barite is usually crushed to a size varying from that of a pea to ½ in., according to the amount of iron oxide, and whether it is in the form of scales or crusts, or as a stain. Jigging is necessary in some cases to remove the iron stain. Higgins[27] has described the bleaching process as follows:

"The crushed mineral is bleached in circular wooden tanks, lined with sheet lead. Each tank is provided with a 2-in. lead pipe, coiled in the bottom, closed at the end, and provided with small perforations, 7 in. apart. Steam escaping from the perforations agitates and heats the bleaching solution. This is sulphuric acid diluted to about 20 or 30° B. The mineral is charged into the tanks to a depth of about 3 ft., after which the acid solution is run in and steam is turned on. The time required to bleach the mineral varies from 6 to 80 hours, depending on the iron content. It is the usual practice to leave the steam on continuously, although good results may be obtained by cutting it off for half an hour at intervals of one hour."

"A tank 4.5 ft. high and 8 ft. in diameter is a convenient size for bleaching, containing, when charged to a depth of 3.5 ft., approximately 14 tons. It should be made of stout, well-seasoned wood, preferably cypress, should be well braced on the outside, and lined on the inside with heavy sheet lead. Connections should be provided so that either steam or water may be supplied through the perforations in the lead pipes. These perforations are best located at an angle of 45° off the vertical rather than directly on top, in order to prevent clogging of the pipes with fines.

"One of the greatest items of expense in the process is the steam for heating and agitating the bath. This may be greatly reduced by use of an ordinary injector,

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[27] Higgins, Edwin: *The Mineral Industry*, vol. xiii, p. 34 (1904).

using a mixture of air and steam instead of steam alone. The amount of air can easily be regulated, so that the acid will not be too much cooled. The temperature may be kept at 200° F.; better agitation will result; the acid will suffer less dilution from condensing steam; and the steam consumption will be reduced one-half."

Should manganese dioxide be present in the barite it is not removed by the bleaching process. It becomes necessary then to grind the barite containing manganese oxide to a paste on a 40-mesh screen, and mix it in the proper proportion with sodium nitrate, salt, and sulphuric acid. The mixture is heated in a specially constructed furnace, which converts the iron and manganese into soluble chlorides that are readily removed by washing, the barite being collected in a series of settling tanks (usually three in number).

After bleaching, the barite is reduced to the size of fine sand by rolls and then to an impalpable powder by buhrstones, after which it is ready to be packed for shipment.

Drying

In drying bleached barite regard should be had for an even temperature, in order that none be sent to the mill imperfectly dried, otherwise it may "sweat" before going to the buhrs, which will yield a finished product that is off color. If mechanical conveyors are used for delivering the barite, care should be taken that the material does not come in contact with oil.

Grinding

For use in the paint and rubber trade the barite must be finely ground. This is accomplished almost entirely by buhrs, since no other method of grinding yet tried has proved so satisfactory in reducing the material to the required degree of fineness.

Barium Compounds

Barium enters as a principal constituent into a number of artificial compounds, which have a wide range of uses. Some of the more important compounds include barium oxide, barium chloride, barium carbonate, lithopone or lithophone, and blanc fixe. These are briefly described below in the order named.

Barium Oxide.—This compound is prepared according to the Bradley and Jacobs process by treating a mixture of barite and carbon in the electric furnace. The following reactions which occur simultaneously are reported:

$$4BaSO_4 + 4C = BaS + 4CO + 3BaSO_4$$
$$3BaSO_4 + BaS = 4BaO + 4SO_3$$

Barium hydrate, used chiefly in the beet-sugar industry, may be obtained from the resultant mixture by treating it with hot water. In France, barium dioxide is used in the production of hydrogen dioxide, which finds an extensive use as an antiseptic and bleaching agent.

Barium Chloride.—As practiced by a Southern plant, this compound is prepared by heating in an 80-ft. rotary cement kiln a mixture of barite and coal, which after reduction yields barium sulphide and barium carbonate. The resulting mixture is leached with water, which removes the barium sulphide, and the residue treated with hydrochloric acid. The sulphide solution is treated with hydrochloric acid, filtered, purified, and evaporated to the point of crystallization.

Barium chloride has extensive use in the color industry and in the manufacture of wall paper. The fused chloride is used to a limited extent in tempering tool steel.

Barium carbonate (artificial), imported as witherite, is used to neutralize the soluble sulphates in those clays employed for the manufacture of terra cotta, to a limited extent in the manufacture of glass, and as a chemical reagent.

The Seurre process[28] for the manufacture of barium carbonate from barite consists in heating to the fusion point with constant stirring equal parts of pulverized barite and calcium chloride with 5 or 6 per cent. of powdered charcoal in a crucible or furnace. The mass when leached in water yields a solution of barium chloride and an insoluble residue of calcium sulphate. Barium carbonate is precipitated from the filtered chloride solution by the addition of a 50 per cent. solution of ammonium carbonate, filtered and washed. The solution containing ammonium chloride may be treated with chalk, and the ammonium carbonate and calcium chloride returned to the process.

Lithopone.—Originally lithopone consisted of a simple mixture of barium sulphate and zinc sulphide, but later this was improved upon and the two were obtained as a double precipitate. At present the product may consist of barium sulphate with either zinc sulphide or zinc oxide or both. In the trade lithopone has been known under different names, such as Charlton White, Orr's White, Griffith's White, Becton White, etc. The best grades of lithopone are white, but inferior grades may be some shade of gray, brown, or yellow, due to the presence of such impurities as carbon, iron, etc.

The method usually employed in the precipitation of lithopone consists in mixing solutions of barium sulphide and zinc sulphate in the proportion of 51 parts of the former to 49 parts of the latter, which results in the precipitation of barium sulphate and zinc sulphide. The barium sulphide is prepared by heating a mixture of crude barite (four parts by weight) and fine coal (one part) to bright red heat in a reducing atmos-

[28] *The Mineral Industry*, vol. xiv, p. 45 (1905).

phere. "In Germany lithophone is made largely from zinc sulphate obtained directly from ore at the works in the lower Harz, but in the United States the zinc sulphate is chiefly obtained by dissolving scrap zinc, or zinkiferous by-products, in sulphuric acid. At one works scrap brass is the source of the zinc employed."[29]

Blanc Fixe is chemically pure barium sulphate, and is precipitated from a solution of barium sulphide either by sulphuric acid or by salt cake. When salt cake is used sodium sulphide is formed, for which there is a limited market. The artificial barium sulphate ("blanc fixe") is pure and free from grit, and is employed chiefly in the manufacture of the finer grades of white paint, oil cloth, and paper. Large quantities of the product used in the United States are imported from Germany.

Until within the past few years this product was confined to the imported lithopone, chiefly from England and Germany, but an important home industry has now been established, and in 1908 at least nine firms in the United States were manufacturing lithopone.

Manufacturers of Barium Products

A table giving the names and addresses of the consumers of crude barite in the United States was published in *The Mineral Industry*, vols. xvii to xx (see bibliography at end of this paper for page references).

Uses

Much the greater part of the production of barite is consumed as an inert pigment in the manufacture of mixed paints, the ground mineral being used both in its natural and in its artificial forms for this purpose. For this use the mineral must be finely ground, washed, bleached with acid, and floated. As a pigment, barite is stable in the presence of acids, alkalies, and gases, since it undergoes no chemical action and is inert as to color. Its crystalline nature renders it unsatisfactory as a pigment if used alone, and only moderate percentages should be used in mixed paints in order to secure the best results. Barite is considered a good base upon which to precipitate other colors, and it is used to dilute other pigments which tend to destroy the oil. Moreover, it gives "tooth"[30] to a coat of paint, so that a second or third coat will adhere.

According to Gardner and Heckel, probably the largest amount of barite is used in the production of the artificial pigment lithopone, which

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[29] *The Mineral Industry*, vol. xvi, p. 95 (1907).

[30] "'Tooth' is an expression used to describe the brushing action of a paint, which indicates that the paint is being taken up by the wood or other surface to which it is applied." *Trans.*, l, p. 985, (1914).

consists of 70 parts of barium sulphate and 30 parts of zinc sulphide. They say:[31]

"An enormous tonnage of this pigment is at the present time being used in the manufacture of interior flat wall paints of the oil type, such paints having almost entirely supplanted the use of corroded white lead which was at one time used for interior painting. The more dense nature, lower price and sanitary value of lithopone were responsible for this change. Lithopone is, moreover, used to a large extent in the manufacture of oil cloth, shade cloth and linoleum. Its whiteness makes it of particular value for this purpose. Barium sulphate is also used of itself in admixture with other pigments in the manufacture of prepared paints. The chemically precipitated form of barium sulphate, which is generally called *blanc fixe*, is also used for this purpose."

These writers conclude that there should be no technical objection to the use of barite in paints. They say:[32]

"The only question is an economic one, and involves two considerations: The first is that barytes and other inert pigments shall not be sold except under their true names and shall not be used as adulterants, using that term in the sense given it by the American Society for Testing Materials, as follows: 'Adulterant—A substance partially substituted for another.' The other consideration is that the amount of barytes shall be such that it will not impair the opacity of the paint under the conditions of application, which implies that three coats shall effectively hide the underlying surface to which it is applied."

The uses of some of the more important barium salts have already been stated. Barium is also used in the manufacture of glass, especially in rolled glass, hollow ware, crystal and table glass, and in such special glasses as the Jena phosphate crown glass, which is reported to contain 28 per cent. BaO. Other uses of barite are in the manufacture of rubber, wall paper, asbestos cement, tanning leather, refining sugar, pottery glazes, enameling iron and oil cloth, poker chips, coating canvas ham sacks, and in the preparation of fertilizers, boiler compounds, insecticides, hydrogen peroxide, and artificial driftwood salts.

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[32] *Idem*, p. 985.

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Depreciation as Applied to Oil Properties

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(New York Meeting, February, 1915)

THERE is a difference of opinion among engineers on the subject of depreciation in general, and still more on its application to any given case The committee which was appointed by the American Society of Civil Engineers to make a report on Valuation of Public Utilities, states "There is no subject connected with valuation about which there are more diverse views than those relating to depreciation;" and in the discussion which took place upon the presentation of the preliminary report of this committee in January, 1914, that part of the report dealing with depreciation called forth more discussion than any other. From this discussion it was evident that the word "depreciation" was not always used in the same sense, so it is quite important that some definition be adopted. As this paper relates to the proper method of treating depreciation by corporations or individuals operating oil properties, both of which must make a return of annual net income to the government in connection with the income tax, it will be in order to adopt the definition of the Internal Revenue Bureau. Under date of Mar. 29, 1910, the Commissioner of Internal Revenue of the Treasury Department stated "Deduction on account of depreciation of property must be based on lifetime of property, its cost, value and use." Again, on Form 1035, "Return of Annual Net Income" for miscellaneous corporations (Section 2, Act of Congress approved Oct. 3, 1913), a deduction from gross income is allowed for "Total amount of depreciation for the year." This item is defined more particularly as follows:

"The amount claimed under Item No. 5 (*b*) for depreciation should be such an amount as measures the loss which the corporation actually sustains during the year in the value of buildings, machinery, and such other property as is subject to depreciation on account of wear and tear, exhaustion, or obsolescence. The amount taken credit for on this account in order to be allowable should be so entered on the books as to constitute a liability against the assets of the corporation. The amount claimed under this item should not cover losses in the value of stocks and bonds. Decrease in the book value of securities owned, so far as such decrease represents a decline in the actual value of such securities, should be deducted under item 5 (*a*) of the return.

"Where depreciation of physical property is made good by renewals, replacements,

repairs, etc., and the expense of such renewals, replacements, repairs, etc., is charged to the general expense account, no deduction for depreciation can be made in the return of annual net income. Where a depreciation reserve is set up, all renewals and replacements must be charged to such reserve and the addition to this reserve each year must be a fair measure of the loss which the corporation sustains by reason of the depreciation of its property."

Here the Internal Revenue Bureau recognizes that depreciation includes wear and tear, exhaustion, and obsolescence, and that in arriving at the amount to be charged off each year there must be taken into account the lifetime of the property, its cost, value, and use. The Bureau also indicates the difference between depreciation and ordinary wear and tear —more or less uniform in amount each year—which is generally carried as a regular operating account under the designation of "repairs and renewals" or some similar title. The Bureau also refers to the setting up of a "depreciation reserve," which is undoubtedly the best way to treat depreciation on the books, where it is carried as a credit account, although on the balance sheet it may be shown either on the credit side as a liability or on the debit side as a deduction from one or more of the property accounts. This account should be credited annually (or preferably monthly) with an amount sufficient to provide a fund to take care of the cost of deferred or final renewals, and in the case of an oil property, to provide a fund to balance the depletion of oil lands. As depreciation reserve is both a debit and a credit account, against it should be charged the cost of deferred and final renewals when made. By so doing the annual net income is not affected by large expenditures made in any one year for new plant to replace that which has been discarded on account of deterioration, inadequacy, or obsolescence. Depreciation reserve should also be charged with the cost of such new oil lands as are necessary to maintain the original value of oil lands. Theoretically, the amount standing in depreciation reserve at any given time plus the value at that time should equal the original value; but seldom does this condition obtain, as it is exceedingly difficult to determine the proper amount to set aside each year in depreciation reserve. Strictly speaking, the only way to arrive at this amount is to make a careful valuation each year, and debit or credit depreciation reserve accordingly. Manifestly such yearly valuation is impracticable, and it is more convenient to assume a lifetime for each item making up the property, and it is this method which is recommended by the Internal Revenue Bureau. It is therefore evident that the annual allowance for depreciation cannot be determined with mathematical accuracy, as both the lifetime and the condition of any property or of its component parts are only estimates, on which no two engineers will agree in detail. It would therefore seem that, particularly as applied to oil properties, any discussion as to the relative merits of the various methods of setting aside an annual amount for depreciation would introduce a

refinement not warranted by conditions. Mention, however, will be made of the three principal methods. By the sinking-fund method a certain equal percentage is set aside each year, which, being compounded annually at a given rate of interest, will at the end of the assumed lifetime amount to the original value. By the equal-annual-payment method certain increasing percentages are set aside each year, which, without interest, will at the end of the assumed lifetime amount to the original value. By the straight-line method a certain equal percentage is set aside each year, which, without interest, will at the end of the assumed lifetime amount to the original value. For oil properties, with an inevitably decreasing production (after maximum is reached), it would seem that some method should be adopted by which decreasing percentages should be set aside each year for depreciation. Such a method would more nearly equalize the depreciation charge per barrel of production. On the other hand, it is a question whether, on account of the higher selling price which generally obtains as an oil field is depleted, the later years cannot stand a higher depreciation charge per barrel. In view of these uncertainties it would seem that the straight-line method is simplest and best, for, owing to the extreme difficulty in arriving at the lifetime of the various items making up the property, the greater exactness called for by the other methods is unnecessary. No matter what method is adopted and no matter how conscientiously and intelligently applied, in all probability the experience of a few years will show that the amount in depreciation reserve is too large or too small, and that the annual rate of depreciation should be changed accordingly. For this reason it is advisable to adopt at proper intervals what has been termed the actual-inspection method, by making a physical valuation of those parts of the property susceptible to depreciation, in order to determine whether or not the amount in depreciation reserve is equal to the original value less present value, as it should be—leaving out of consideration the effect which appreciation and other elements connected with valuation may have on the subject, and assuming that original cost and original value are equivalent terms.

While the proper treatment of depreciation is required by the government for purposes of taxation, it is of still greater importance to the stockholder, for only then does he know that his investment is being kept up to its original value, and that the dividends paid, whether 10, 50, or 100 per cent., are really earned. Whether depreciation reserve is invested in outside securities, as may be advisable in the declining days of an oil property, or whether invested in the property itself for betterments and additions, is immaterial. If properly set aside, the stockholder knows that out of earnings each year has been provided a fund, not applicable to dividends, which maintains his investment at a constant value, and which, when the oil is exhausted, will be sufficient, together with the value of the remaining property, to pay him back his principal. This of course repre-

sents an ideal condition, seldom realized in practice; for while it is very easy to enunciate principles, it is most difficult to apply them to the particular case in hand. Such application, however, will now be attempted.

First of all, the lifetime of every item entering into the property should be established, and a depreciation reserve set aside annually for each item based upon such lifetime, or an average lifetime established for the entire property. An oil property, the lifetime of which as a whole, or in part, is so uncertain, may be considered as being made up of three classes, viz.: oil lands, field equipment, and wells. In field equipment should be included tanks, reservoirs, pipe lines, loading racks, water system, power plants, structures of various kinds, and miscellaneous items belonging to the field as a whole rather than to the individual well. In wells should be included the well itself and those appurtenances, the lifetime of which depends more upon the individual well than upon the field. This simplifies the matter in that only the average lifetimes of the oil lands, the field equipment, and the individual well need be established, thus reducing book entries to a minimum.

It is manifestly beyond the province of this paper to select any particular oil property for the purpose of illustrating the general principles of depreciation, and even were such property thus selected, the application of those principles to that particular case would have no more value when applied to another property, than if the field as a whole or even a State as a whole, such as California, were selected for purposes of illustration. It is obvious that a rate of depreciation established for the property containing the Lakeview gusher, for example, would be of little value when applied to another property in the same field, or even to an adjoining property. For this reason the State of California as a whole has been selected for purposes of illustration, and it is believed that the end in view will be better served than if some specific property were taken. In establishing the lifetime of the oil fields of California as a whole or in part, the best recourse is to the United States Geological Survey, which for years has made a careful study of the various oil fields of the United States and publishes a report every year upon the subject. For California the Survey has estimated that the original contents of the probable oil lands were somewhere between 5,000,000,000 and 8,500,000,000 barrels, or from one-third to one-half of the entire probable production of the United States. There have been produced in California as a total to date some 750,000,000 barrels, leaving still in the ground from 4,250,000,000 to 7,750,000,000 barrels. At the present rate of production, 100,000,000 barrels yearly, it will take from $42\frac{1}{2}$ to $77\frac{1}{2}$ years to exhaust the oil, with an average of 60 years. It must be remembered, however, that within the past 10 years the annual production of the State has trebled and within the last five has doubled, the production in 1904 having been 32,743,273 barrels and in 1909, 54,433,010 barrels.

On account of this constant annual increase in production, as well as the danger from water intrusion, which seriously threatens some fields of the State, it would seem prudent to adopt a lifetime of not more than 25 years for the field; and assuming that the oil lands owned by the hypothetical company under consideration are up to the average, a depreciation rate of 4 per cent. per annum is indicated upon their cost, in this case considering cost and value as equivalent terms. This annual rate, being predicated upon total extinction of value in 25 years, will be modified in accordance with the value of the land after the oil is extracted. For example, should the land after the oil is extracted be worth 25 per cent. of its original value, the yearly depreciation rate will be 3, instead of 4 per cent. For purposes of illustration, however, the land, upon exhaustion of the oil, will be considered as of no value, which is not far from actual conditions in some parts of California. In order to apply the principle, some cost (value) of oil lands must be assumed, and for purposes of illustration a cost of $500 and $1,000 respectively per barrel of daily production will be used. On a production of 1,000 barrels per day, the cost will be $500,000 and $1,000,000 respectively, with yearly depreciation charges of $20,000 and $40,000 respectively, which on a yearly production of 365,000 barrels amounts to $0.055 and $0.110 respectively per barrel. As depreciation charge on oil lands takes the place of royalty paid by companies operating under lease, the results obtained are not unreasonable.

In order to establish the lifetime of field equipment, that of each component part must be estimated and a general average established. Some of the equipment may last the lifetime of the field. Other parts may have to be renewed every five years, so that here again each property must be studied by itself. Assuming an average lifetime of about 15 years, which seems not unreasonable, an annual depreciation rate of 7 per cent. is indicated. In applying this rate to a company of which I have knowledge, with a field equipment costing $150,000 per 1,000 barrels of production daily, the annual depreciation charge will be $10,500, or $0.029 per barrel. In applying to another company of which I have knowledge, with a field equipment costing $270,000 per 1,000 barrels of production daily, the annual depreciation charge will be $18,900, or $0.052 per barrel.

In regard to the lifetime of an individual well, that again is a local question and must be determined in accordance with facts to be ascertained in each particular field. There are wells in the Kern River district which have been in operation for 15 years and are still producing a fair percentage of their original production, but this is an exceptional field for persistence. In this determination may also be considered the cost of drilling additional wells in order to maintain production at a constant figure. This is frequently carried as a regular operating expense, similar to repairs and renewals, but as depreciation, as herein defined, is a function of capital account rather than of annual production, and as such expense may vary greatly from year to year, it is a question whether drilling

to maintain production should not be charged into capital account, and depreciated along with the other wells. Such a treatment naturally increases the annual percentage of depreciation, but as this rate is, after all, a matter of estimate and subject to change as new conditions develop, it seems preferable to treat drilling to maintain production as a charge against capital rather than as an ordinary operating expense. Giving due consideration to this element in the problem, 10 years will be assumed as the average lifetime of a well and its appurtenances, taking also into account dry holes, having no lifetime, on the one hand, and those wells which may produce for 20 years or more on the other. Applying this annual depreciation charge of 10 per cent. to one company of which I have knowledge, with an investment of $190,000 in cost of drilling wells per 1,000 barrels of production daily, and to another with $260,000 so invested, there will be a yearly depreciation charge of $19,000 and $26,000 respectively, amounting to $0.052 and $0.071 respectively per barrel.

Summing up these depreciation charges we arrive at the following results per barrel of production:

	Per Barrel
Depreciation on oil lands (royalty)	$0.055 to $0.110
Depreciation on field equipment	0.029 to 0.052
Depreciation on wells and appurtenances	0.052 to 0.071
Total depreciation	$0.136 to $0.233

These figures must not be regarded either as the minimum or the maximum which may obtain in any given operation, and particularly is this true of oil fields outside of California, and where the persistence of the wells may be greatly different. These figures, however, serve to call attention to the important bearing which a proper charge for depreciation has upon the cost of producing oil in a State where, during the past few years, prices at the well have ranged from 30c. to 85c. per barrel, depending upon the gravity of the oil and the location of the field. It must also be remembered that, in addition to this charge for depreciation, there will be an item for renewals and repairs, charged directly into the cost of operating, which may range from 3c. to 5c. per barrel. It is therefore evident that the actual cost of producing oil in California, no allowance being made for interest on the investment, must in many cases approximate the selling price, as shown in the following statement, based upon experience:

	Per Barrel
Pumping	$0.04 to $0.05
Miscellaneous field expense	0.04 to 0.06
Repairs and renewals	0.04 to 0.05
General expense	0.02 to 0.04
Total direct cost	0.14 to 0.20
Depreciation as above	0.14 to 0.23
Total cost	$0.28 to $0.43

That a proper depreciation charge may be equal to, or greater than, the direct cost per barrel will doubtless surprise many investors in oil properties who consider only immediate expense and immediate profit without regard to the safety of the principal invested.

To engineers, however, who have had the responsibility of operating oil properties, these figures will not be surprising; for in his interesting and valuable paper on Present Conditions in the California Oil Fields, presented to the Institute at its San Francisco meeting in October, 1911, M. L. Requa states:[1]

"It is exceedingly to be regretted that the oil-producers of California, as a whole, do not apparently realize the real cost of production. . . . From territory of, say, 2,500 ft. depth, total costs will approximate from 30 to 35 cents per barrel. For direct production—*i.e.*, pumping, cleaning, and pulling—10 cents per barrel may be safely assumed. For maintenance of surface-equipment and rigs, 4 cents is a conservative estimate. For exhaustion of oil-land, and redemption of capital, from 6 to 10 cents must be reckoned; and for drilling to maintain production, 12 cents is not excessive. These figures make a minimum of 32 cents and a maximum of 36 cents."

While Mr. Requa has arrived at his estimated cost by a different method of accounting, the average is practically the same, and it would be most instructive to have the opinion of other engineers on this subject. To draw attention to the proper application of depreciation to oil properties, and to its importance as an element in the cost of producing oil, is the purpose of this paper, and it is hoped that a full discussion will be made by those members of the Institute whose experience in the various oil fields of the world will be of great value, not only to engineers, but also to all those interested in the development of oil properties.

To sum up, my conclusions are:

That there should be set aside each year, out of earnings, a depreciation reserve, sufficient in amount to keep the property at a constant value, by providing a fund to take care of extraordinary wear and tear, inadequacy, obsolescence, and exhaustion.

That ordinary wear and tear should be charged directly into operating.

That depreciation reserve, being both a debit and a credit account, should be charged with extraordinary repairs and with replacements as made.

That depreciation reserve should not be applicable to the payment of dividends, but only for the repayment of principal.

That depreciation reserve may be invested in the property itself or in outside securities.

That the principles used in setting aside depreciation reserve should be those outlined by the Internal Revenue Bureau, quoted in the beginning of this paper.

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[1] *Trans.*, xlii, p. 841 (1911).

That in the application of these principles it is necessary to determine the lifetime of each item making up the property, but that, in view of the many uncertain factors pertaining to oil properties, it is sufficiently accurate to divide depreciable property into three classes—oil lands, field equipment, and wells—and determine the average lifetime of each class.

That these average lifetimes, with due consideration of remaining value upon exhaustion of oil, will determine the annual percentage to charge on these three items standing in capital account, the proceeds of which should be credited to depreciation reserve.

That these percentages, being based upon estimates, are subject to change, and should be verified at proper intervals by making actual valuations in order to determine whether the amount in depreciation reserve plus present value equals original value, as it should—neglecting other elements of valuation, and using cost and value as equivalent terms.

That in establishing lifetimes each property must be studied by itself, and different annual percentages must necessarily be adopted for different properties on account of the great variation in conditions.

That it would, however, be advantageous to establish for each oil field average percentages which could be used until such time as the characteristics of the different properties in the field are known.

That for California such average annual percentages should be 4 per cent. on cost of oil lands, 7 per cent. on cost of field equipment and 10 per cent. on cost of individual wells and appurtenances, but that these percentages should be changed in accordance with the development of the characteristics of the property under consideration.

That in the oil fields of California depreciation cost may equal or even exceed direct cost of producing oil per barrel.

These conclusions are not intended to be dogmatic, but rather for the purpose of stimulating discussion, for only by combining the experience and judgment of many engineers can depreciation be correctly applied to oil properties.

Discussion

C. E. Grunsky, Jr., San Francisco, Cal. (communication to the Secretary*).—The difference of opinion of engineers on the subject of depreciation is justified, to some extent, by the many viewpoints from which such a subject can be considered when applied to properties of widely differing physical characteristics.

Mr. Henry takes up the subject of depreciation as applied alone to oil properties, and in doing so makes a clear distinction between amortization and the repair or renewal requirement.

After carefully reading the paper the purpose of its author in suggesting a "depreciation" reserve or fund to keep the property at a *constant*

* Received Feb. 1, 1915.

value does not seem clear. As shown by Mr. Henry, the depreciation element as compared with the operating expense per gallon of oil is very large because of the comparatively short life of the oil properties, and it is also evident that such a "depreciation" fund would in a short time contain a very large amount of money.

It seems desirable and permissible in income-tax matters to take into account the depreciation of property and to separate from the income such money as is necessary to amortize the invested capital to an extent warranted by the depreciation. However, it does not seem either necessary or desirable to create a large fund which will in time approximate the entire invested capital and which may be invested by the directors, as Mr. Henry suggests, "in the property itself or in *outside securities*."

This method seems cumbersome and liable to offer in the reinvestment of such a fund temptations to a directorate that might show more devotion to its own interests than that of the stockholders.

A reserve fund is very frequently desirable in mining and oil operations, but to keep up a fund of this magnitude or to reinvest it in outside securities does not seem desirable from a stockholders' standpoint. The money withdrawn from the receipts to be placed in the depreciation fund (not including replacements and the drilling of new wells necessary to keep up the productiveness) should be returned to the stockholders of the company, not as dividends, but as an amortization of the invested capital.

Past practice has been to return the invested capital as dividends, no differentiation being made between profits and returned capital, and no provision being made for any fund called a "depreciation" fund, or what you like, for the purpose of amortizing the capital. During the life of a property (annually if possible) all of this depreciation fund should be returned to the stockholders, excepting such portion as it is desirable to retain intact to insure the continuity of operations, and in such event outside investment from this fund by the directors should be prohibited.

In Form No. 1031 (revised), *Return of Net Annual Income—Corporations*, issued by the Internal Revenue Bureau, under the heading "5 (B) Depreciation and Depletion," the following limitation is placed on the amount to be deducted from income because of the exhaustion or the depletion of natural deposits: "In computing depletion in the case of natural deposits the rate should not exceed 5 per cent. of the gross value at the mine of the ouput for the year."

This requirement determines that the total amount deducted from the gross income during the life of the property, which is to be laid aside or returned to the investor as capital, shall not be more than $1/20$ of the gross value of the total production from the mine or oil property. If, under this ruling, the owners or investors pay a large sum of money for the property or erect an expensive plant because they can recover the prouct at a low cost per ton or per barrel and the investment exceeds this

$\frac{1}{20}$ of the gross value of the total production, a portion only of their capital will be returned as dividends with the income tax subtracted from it.

The proper way to estimate this "depletion" is, as pointed out by Mr. Henry, to use as a basis for the estimated life the average life of the oil properties in the vicinity of any property. Then the annual depletion will be the annual installment which is necessary to return to the owner his legitimate or proper investment during that period of years.

When it is observed that the actual life of an oil property will be prolonged beyond that originally expected, the estimated life can be increased, and when it is seen that the output is decreasing rapidly, the reverse is possible. Under no circumstances, of course, should the owner be allowed to credit, because of the depletion of his property, more money to the depreciation fund than has been legitimately invested by him.

The fact that the remaining life of the properties may be extended from time to time is discussed at some length by C. E. Grunsky in a paper published in the *Transactions of the American Society of Civil Engineers*.[2] Those who care to go more deeply into this phase of the question may refer to this paper.

I. N. Knapp, Ardmore, Pa. (communication to the Secretary*).—As a general accounting proposition on depreciation, Mr. Henry's paper is undoubtedly correct. In discussing his paper I can only give my personal experience and opinion from the standpoint of an oil and gas operator.

The depreciation on buildings, machinery, tanks, and pipe lines may be readily agreed upon, as they are on the surface and are easily examined as to their condition from time to time, and a civil engineer used to such public utilities would be entirely competent to pass on that part of the problem.

A government ruling as to depreciation or losses is too often a "heads I win, tails you lose" proposition which may have to be referred to the courts to tell what is meant. Such rulings may be applicable to the collection of taxes, but do not really tend to show whether any particular property or business is paying or not, or what may be the real depreciation.

To be properly qualified to estimate on the depreciation of oil and gas producing wells, which necessarily reach many feet below the surface,

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* Received Mar. 2, 1915.

[2] Depreciation as an Element for Consideration in the Appraisal of Public Service Properties, *Transactions of the American Society of Civil Engineers*, vol. xl, No. 9 (Nov., 1914).

requires knowledge that can be acquired only by actual experience in drilling and operating.

The general public is very much misinformed as to the actual costs and depreciations of oil and gas properties. The fakir, the boomer of fraudulent oil and gas promotions, the space writer for the magazines, all have spread abroad a sort of idea that all that was necessary to get a big-paying property was to hire a driller to put a few pipes in the ground most anywhere and tap vast stores of oil and gas provided by nature and get a perennial yield of oil and gas at a slight cost and no depreciation.

These ideas are reflected in legislation and decisions of the courts.

To assist the oil and gas producer to meet unwise legislation and imposition of unreasonable taxation and bureaucratic interferences, the producer must inform himself as to what the itemized costs of producing are and what the depreciation is on his properties.

Some years ago I operated in a small oil pool and kept my accounts according to a classification outlined by an accomplished accountant.

A record of all bills paid and the items received was sent in for monthly audit and a trial balance returned, thus giving a monthly check on the business.

Once a year a complete inventory was made, using the same measure of value for each producing well, and the depreciation for the year worked out. The routine work was done by clerks, but the trial balances and inventory sheets were checked and signed by the accountant.

We found the depreciation in yield on about 60 oil wells regularly operated was 3 per cent. per month, or about 28 per cent. per year. The total cost (including all proper charges and depreciation) of putting the oil into the field tanks was 53c. per barrel for a period of years.

The yield from gas wells was so erratic and measurements so uncertain that I could never get a satisfactory line on the cost of production or depreciation. I regret that the books of this operation are no longer in existence, as they gave costs and depreciation in detail.

The wells were pulled out when from five to seven years old because oil fell to 27c. per barrel; it had been as high as $1.38.

The operation had already paid out and made considerable profit, but our books showed that there was no margin at 27c. for continued operation.

Possibly it would have paid to have kept the property for a rise in the price of oil. The reason for not doing so was the probable excessive depreciation of the property and wells when not regularly operated, the heavy taxation put on oil and gas properties; also the majority of the land owners insisted on continuous operation.

The Use of Mud-Ladened Water in Drilling Wells

BY I. N. KNAPP, ARDMORE, PA.

(New York Meeting, February, 1915)

Introduction.—The special object of these notes is to describe the mixing, testing, and use of mud-ladened water for rotary drilling in such a way as to make them helpful to the driller, the operator, or the engineer in solving his own special drilling problems.

The structures, apparatus and tools used are indicated in a general way. No attempt is made to describe the art of rotary drilling; only such descriptions as are necessary to make plain the use of mud are given.

The information is the result of actual experience in drilling in Coastal Plain formations. The materials encountered in the wells drilled were unconsolidated sands, gravels, and clays, in which thin layers of sandstones, shell conglomerates, and shales began to appear at about 1,200 ft. in depth, although one well was drilled to 3,018 ft. without encountering any cemented or indurated materials.

Unusual Conditions.—The general surface of the ground where the drilling was done being hardly a foot above mean tide, and the daily tidal variation being about 15 in., it was found necessary to plank over the sod surface with 3-in. plank to work from. A 30 by 30 ft. planked area was sufficient to carry the derrick with equipment and 2,500 ft. of 4-in. drill pipe stacked in it. The derrick was set up 3 ft. on cribbed blocking arranged to spread the load over the planking. The engine, boilers, and pipe yard were also carried on 3-in. planking.

Drilling Outfit.—This consisted of a derrick, 20 by 20 ft. and 84 ft. high, two 35-h.p., "oil country" boilers, one double 8 ¼ by 10 in. reversible engine, two 10 by 6 by 12 in. duplex mud pumps, one 15-in. rotary, "hoisting works" with chain drive, hoisting block with line, crown block with pulleys, rotary jetting swivels, hose, complement of tools, pipes and connections, and a mud mixer, with engine.

Hydraulic Rotary Drilling Method.—This method of drilling is an adaptation of Fauvelle's[1] invention made in France in 1833 for drilling

[1] See *Journal of the Franklin Institute*, 3d ser., vol. xii, pp. 369 to 372 (1846), for abridged translation of Fauvelle's own account On a New Method of Boring for Artesian Springs.

artesian wells, and was the progenitor of the water-flush method now employed in Europe and the hydraulic rotary method of America. It is absolutely necessary to use this method to complete a well to depths of 200 ft. and over in the Coastal Plain formation. Its success depends on the proper use of mud-ladened water. Rotary drilling is an art that necessarily requires skillful work, and the best results in any district can only be had after numerous test wells have been drilled, the water sands and productive oil or gas horizons located, and proper drilling and casing setting methods worked out.

Initial Ground Pressure.—This in any oil or gas field is in general less than the hydraulic head of 43 lb. per 100 ft. of depth.

Mud-Ladened Water.—Water can easily be ladened with any clayey material that will remain in suspension for a considerable period of time. Such mixture may weigh 33 per cent. more than the same volume of clear water, or give a head of 57 lb. per 100 ft., or a margin of safety of 14 lb. on each 100 ft. of depth. Heavier or lighter mixtures can be used, but that mentioned was found to give good average results.

The time of settling of clayey material in still water varies greatly. One mixture will settle clear in a week, another in six months.

Laboratory experiments show that matter in suspension imparts a certain viscosity to fluids and the rate of settling is by no means dependent solely upon the difference in specific gravities, or upon the fineness of subdivision of the solid.[2]

Requirements.—It is fundamental when drilling by the hydraulic rotary method that a circulation of mud-ladened water, such as described, be maintained from the surface down through a vertical rotary drill pipe and bit, and up between the drill pipe and the wall of the well back to the surface. Moreover, at all times, the mud pressure at any point in the well must exceed the pressure in the strata penetrated at that point, thus excluding all gas, oil, and water as they are encountered. The proper use of mud-ladened water gives a means of drilling wells and keeping them in a state of equilibrium until screens can be set in productive horizons and casing set and cemented in. Thus complete control is given in bringing them to producing. As a practical matter, in a drill hole 1,500 ft. deep a good mud will not settle in a week sufficiently to prevent a drill pipe from being run freely to bottom, and the mud pump from starting circulation, so rotating and drilling may be resumed. Also, such mud will not stick or freeze in a drill pipe if left in 24 hr.

It was necessary to use a power-driven mixer to supply, and a mud pit

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[2] *American Chemical Journal,* vol. xlvi, p. 278 (1911) on The Viscosity and Fluidity of Suspensions of Finely Divided Solids in Liquids. Also, U. S. Geological Survey *Professional Paper No.* 86, Transportation of Débris by Running Water, p. 228 (1914).

to store, sufficient mud properly to control a well drilled in the formations named. Such a storage pit is formed on the surface by building a dike about 3 ft. high around an area 20 by 30 ft. from which the sod has been taken. It must be so located that the thick mud maintained in it can be quickly drawn out into the pump pit (hereafter described) in quantity as desired to thicken the circulating mixture. Mud mixers of suitable design can be purchased of the well-supply houses.

Mud Tests.—It is very difficult to judge whether any particular clayey material will make a suitable mud. The only sure way to know is to mix a sample of available material and try it.

Mud can be best tested by weighing. I used a common market beam scale (without pan) having a capacity to weigh 50 lb. by ounces, and a common galvanized iron bucket in which to weigh the mud. Four equally spaced 1/4-in. holes were punched under the top rim to furnish level marks to fill to.

The bucket is first hooked on under the beam, the poise brought to zero, and the whole balanced by counter-weighting. The bucket is then filled to the level holes with clear water and its weight taken and noted. It may be, for instance, 18 lb. The poise is again placed at zero and more counter-weight added to balance the clear water. This water is then thrown out and the bucket filled with mud, and the poise, when balanced on the beam, will indicate the additional weight of the clayey material in suspension over that of the same volume of clear water. The bucket may be filled with mud as many times as desired and weighed without further adjustment. If the additional weight found at any time should be 6 lb. it would indicate the mixture weighed about 33 per cent. more than the same volume of clear water. If a good working mixture is found to be 5 lb. of additional weight per bucket of mud, and in drilling it increases to show 6 lb., clear water may be slowly added until the weight drops to 5 lb. or a little less. This weighing is simple and practical and is no fad. It is just as necessary for the driller to have a scale and know the weight of the mud he is using as to have a gauge to show the mud pump pressure. A well can be drilled without a pump gauge, or scales to weigh the mud, but it can be drilled safer and in better time if both measurements are intelligently used.

Other Tests.—A water circulation carrying a considerable percentage of sand in suspension is not a proper mud for rotary drilling, although it may meet the weight test.

If for any reason a circulation of this kind is stopped, the drill pipe is likely to "freeze in" in a short time, depending on the rapidity with which the sand may settle. The amount of sand in suspension may be approximately estimated by taking one-third of a bucket of such mud mixture and adding one-half of a bucket of clear water, stirring thor-

oughly, and letting stand half an hour. If less than an inch of clear sand then shows in the bottom of the bucket the circulation is reasonably safe to use. If it is much more it may not be safe. This is a matter that only experience and good judgment can determine.

The percentage of sand in any mixture can be accurately determined with a hand centrifuge with two-arm sedimentation attachment and graduated glass tubes. A proper mud shows no sign of separation—except perhaps 1 or 2 per cent. of sand—if put into such a machine and

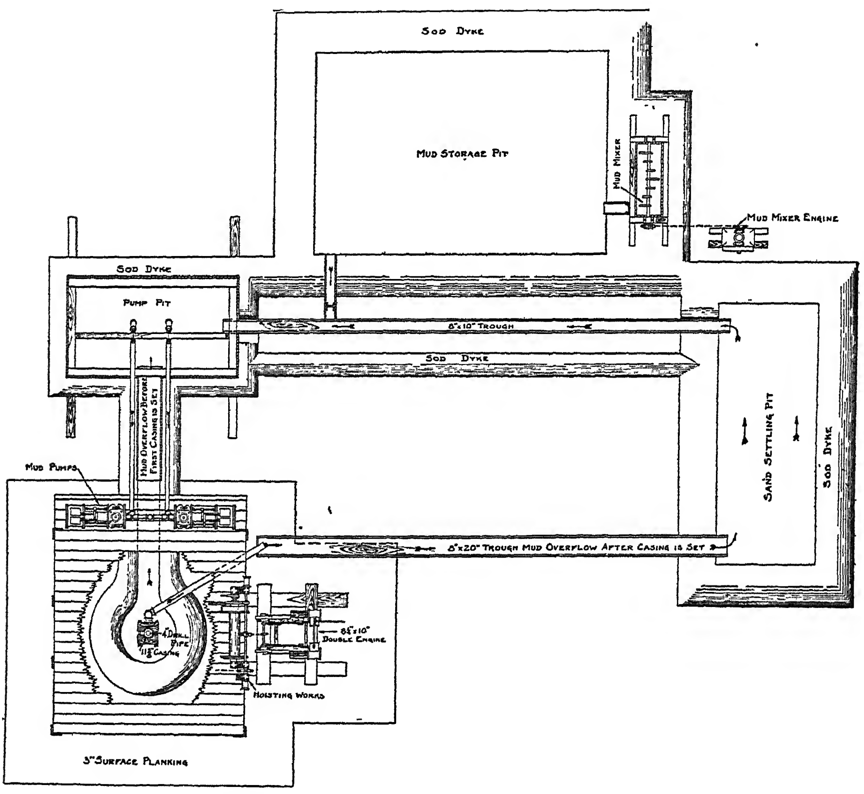

FIG. 1.—PLAN OF WELL-DRILLING PLANT SHOWING CIRCULATION OF MUD.

run at 1,200 rev. per minute for 5 min. This shows that the precipitation of suspended clayey matter in water is extremely slow and that sand settles very much quicker. By mixing mud half and half with saturated brine a quick separation takes place. Turning the machine for 2 or 3 min. at the speed mentioned produces a complete separation of all suspended matter. The precipitation shows itself in layers in the graduated glass tube, the sand coming down first.

Pump Pit.—This pit is located on the pump side of the derrick, with its longest center line parallel to and about 20 ft. from the derrick base.

(See Figs. 1 and 2.) A convenient size is 8 by 16 ft. (inside timbers) by 5 ft. deep, and lagged with 2-in. plank 7 ft. long, placed vertically on the outside of a square set of 8 by 10 in. timbers.

The 2 ft. of lagging above the ground is backed with sod to prevent surface water from running into the pit. A sod dike about 6 ft. in diameter, with the well location as a center, and 20 in. high, is built on the 3-in. plank. This is connected directly to the pump pit by a ditch

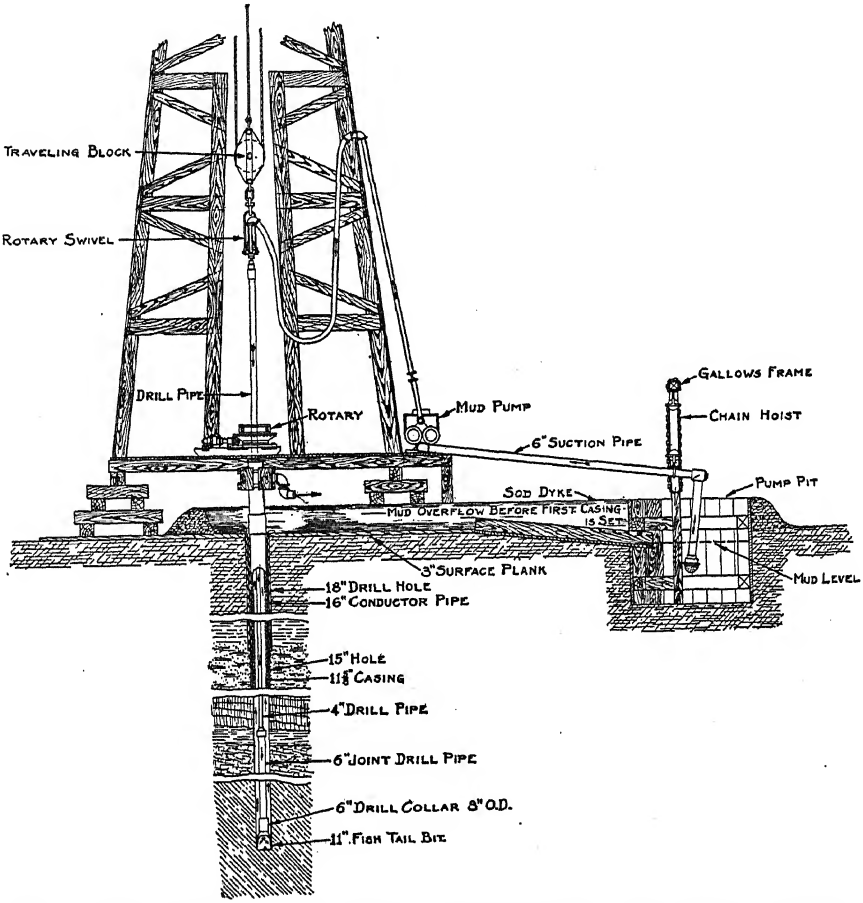

FIG. 2.—ELEVATION OF WELL-DRILLING PLANT SHOWING CIRCULATION OF MUD.

banked up 20 in. by 2 ft. wide. The dike protection was absolutely necessary, as the tide rose at times as much as a foot over the surface.

Pump Strainer.—The large amount of roots and trash in the clay available for mud mixing, and also the fact that wood and swamp materials were drilled through at various depths, made it necessary to arrange for frequent cleaning of the pump strainer. The best arrangement found for conveniently raising and lowering the pump suction pipe for this purpose

was a 1-ton chain hoist hung over the pit from a gallows frame, as indicated in Fig. 2. With it one man could handle a pump suction with ease while removing floating trash from the strainer with his hands.

Preparations for Drilling.—The drilling apparatus being all ready, a mud pump is started and the thin water pumped out of the pump pit. The latter is then filled with mud to above the ground or tidewater level, so there can be no chance of the inflow of clear water thinning the mud. A circulation is then established, by operating a mud pump, from the mud pit through the drill pipe and bit, back to the pit, and continued until the whole content of the pit is properly tempered.

Drilling.—The rotary is started and the drill begins at the surface to make hole. To a depth of 15 ft. I used an 18-in. fishtail bit with 6-in. shank and 6 by 10 in. drill collar (12 in. O. D.) on one joint of 10-in. pipe, followed by a string of 6-in. drill pipe. Near the top of the hole slabs of mud as large as a man's two hands were cut up by the drill and floated out, and had to be shoveled away. Also mud was lost to crawfish holes for a time before they became mudded off.

A 16-ft. length of galvanized, corrugated, culvert pipe was used for the conductor pipe and driven into place. (See Fig. 2.) The top was belled out and spiked to the 3-in. surface planking.

The 18-in. fishtail bit being replaced by a 15-in. bit, the drilling is continued through the conductor pipe and the mud kept close to some standard weight. The large drill collar (12 in. O. D.) on a 10-in. joint of pipe in a 15-in. hole retards to some extent the rate of drilling in unconsolidated material. On the other hand, its use compels the drilling of a round straight hole that requires no reaming before setting a string of casing; also, its use seems more effectively to plaster the drill cuttings and mud into the walls of the well in puddling off the porous strata as encountered, and so preventing loss of mud. In pulling out, there is no dragging or turning of the drill pipe, and in running in, the bit goes plumb to bottom.

As a matter of experience, it had been found that a well drilling in soft sandy material with a 15-in. fishtail bit, followed by a 6-in. (about 7-½ in. O. D.) drill collar and 6-in. pipe, was easily deflected by concretions or pieces of wood, resulting in crooks or humps in the hole. In extreme instances, after pulling out and starting in with a sharp bit such crooks or humps would form enough obstruction so that the bit would not spud past them, and upon rotating a new hole would form, side-tracking a considerable depth already drilled, which was consequently lost. Furthermore, it was often necessary in running back into crooked holes to start pumping and rotating before getting to bottom, because materials which were scraped off the walls of the well and carried down made a bridge or stoppage. In any case a crooked, humped, or deflected hole requires the

use of a rotary shoe or reamer to size the hole down before a string of casing can be run in and set.

Just before pulling out the drill pipe, the mud in the pump pit and the surface circulation should be stirred up, so as to weight the descending mud in order that the drill string will uncouple dry. In pulling out, mud must be frequently pumped into the well so that the "head" of mud, or pressure from within the well, will meet the fundamental requirement before stated. After running a drill pipe to bottom or to a bridge in the hole, no attempt to start a circulation for drilling should be made until the surface mud is first circulated and brought to a proper temper.

In case of a heavy rain it is best to stop drilling and pumping, and pull up off bottom before the circulation thins out. With a good mud there is no danger of "freezing in" even if the drill pipe is left in over night. A circulation should be maintained only when rotating. Pump pits and mud circulation must always be protected against surface drainage and rise of tide.

Action of Water and Mud.—Clear or turbid water flowing on the surface will erode unconsolidated materials, the rate depending upon the velocity or grade and upon the volume. It has also a solvent or slacking action on earthy matter. If such flow was, for instance, through a shallow ditch it might perhaps cause the banks to cave sufficient to make a meander outside of the dug channel. If, however, clayey matter is added to the flowing water the erosive action weakens until a point is reached where erosion practically ceases and the suspended matter will begin to build up and protect the loose material. The solvent action also will be very much weakened and in such a case a shallow ditch in loose material will tend to keep its shape as dug. Clear water will continue to sink in sand or porous material in quantity for an indefinite time. Turbid water will gradually render sand or porous material impervious, but the suspended matter will be carried in and deposited in the sand to a considerable depth. A good mud (or mud-ladened water) will render sand or porous materials impervious almost instantly and the penetration of the mud is small. These are all matters of common observation.

If clear or turbid water is circulated in rotary drilling such water will erode the walls of the well, particularly in loose sands and sandy materials, and cause them to cave and to lose their cylindrical form.

Sticky clays and such unconsolidated strata as can resist the erosion of circulating clear water will drill much faster with it than with mud, on account of its solvent action. It is a matter of necessity, however, to start with and keep a mud circulation of a standard that will resist erosion of the weakest sand strata, if the well is to be kept in proper cylindrical condition for successful cementation and completion. If the walls of a well in sand or other material cave, a large amount of such

material will come out with the overflow. This caving increases enormously the area to be mudded off.

Gain or Loss in Mud.—In rotary drilling there is always a slow increase in the weight of the circulating mud. When the mud gets too heavy a part of it may be pumped back into the storage pit and clear water slowly added to the circulation until the original volume and standard weight are restored.

In drilling a 15-in. hole through sand beds the amount of mud lost should be observed. For instance, in my drilling through 60 ft. of coarse reddish water sand the pump pit would have mud from storage drawn into it several times, equal in all to about 4 ft. in depth by 9½ by 17½ ft. area, or 673 cu. ft. If this mud is one-third to one-half material in suspension, there would be 226 to 337 cu. ft. of material to lodge in the pore space in the sand. The porosity of sand ranges in general from 26 to 47 per cent., so it is fair to assume 38 per cent. as an average. If we assume that the mud may penetrate the walls of the 15-in. well 2 ft., we have a column of sand 5 ft. 3 in. in diameter, less the 15-in. core, and 60 ft. long, or 1,225 cu. ft., having a porosity of 456 cu. ft. to hold suspended material. When the mud was properly mixed and skillfully used, the loss to the porous strata was found to be surprisingly small, and, on the other hand, the amount of cuttings brought out was also small and did not appear to exceed the cubical contents of the hole which had been made.

In my first attempts at drilling in this formation too much clear water was sometimes used, resulting in caving from several sandy horizons at the same time. At such times as many as four men were busy shoveling away the sand that was being brought out in the overflow, and in such cases it required the use of much mud to stop the caving. At these times there was also danger of "freezing in" the drill pipe.

The Sampling of Drill Cuttings.—It is extremely difficult in rotary drilling to get reliable samples of the material passed through. If clear or turbid water is used, the cuttings tend to dissolve or separate by the jigging action of the ascending circulation, and the heavy particles lodge in the caves and erosions in the walls of the well and mix with the caved materials. It is very difficult in such cases to find any material in the overflow from which to judge the strata through which the drill has just passed. With a heavy mud there is much less dissolving, mixing and separating of the drill cuttings, and, on washing out a sample of mud, particles may be found which fairly indicate the material just drilled through. I found that a No. 18 mesh bronze wire mosquito screen set in a wood frame about 16 by 20 in. made a good screen on which to separate the shells and coarse particles from the mud by washing. Gas sands can be recognized by their porosity, as revealed by examination under a magnifying glass of 20 diameters. Oil sands have the same kind of porosity and in addition can be recognized by their taste.

After setting and cementing[3] in a casing and using a regular mud, in drilling out a cementing plug it was observed that wood cuttings appeared in about 25 min. and brass, from a back-pressure valve, in about 40 min., when the calculated flow would have taken 30 min., showing the acceleration by floating of the wood and retardation by sinking of the brass.

Open Hole.—With a good mud and skillful working there was no difficulty in drilling 700 ft. of 15-in. open hole before setting the first string of casing ($11\frac{5}{8}$ in.). The drill passed through several layers of fine water sand, having a total thickness of perhaps 200 ft.; through 100 ft. of sandy materials and shells, and 400 ft. of clays and sticky mud. The short ditch, about 25 ft. long and the pump pit, previously described, were of sufficient settling capacity, because the proper use of mud did not permit the sands to run. Starting with the storage pit full of thick mud, it was not found necessary to mix any additional mud to reach a depth of 700 ft., under the conditions named. As a matter of experience, I drilled 1,880 ft. in depth of $8\frac{1}{2}$-in. hole with the rotary, using mud, going through three well-known water-bearing loose sands with strata of clays, shales, shell beds and some sheets of rock, and then set a string of $6\frac{5}{8}$-in. casing, 60 ft. of 14-in. O. D. drive pipe being all the other pipe in the hole. There was no caving and but little loss of mud. The work was done in day time only and the drill pipe was many times left in over night with no sign of sticking or freezing.

First String of Casing.—This should set so that the top coupling will come just above the 3-in. surface plank, and so that a drilling nipple can be screwed into it. This nipple should have a 6-in. hole cut in its side, so as to bring the overflow from the well as high up as possible under the derrick floor. A wood block is fitted to match over the side hole and a 6-in. pipe flange bolted to the latter and the whole clamped to the nipple. A 6-in. pipe is connected (with nipples and two elbows to make a swing) to the flange to carry the mud circulation of the derrick to an overflow just back of the place where the driller stands at the hoisting-drum brake. The advantages in bringing the overflow up to this point are that it can easily be seen by the driller and that there is a better chance for getting samples of cuttings than from a ditch. Flash boards may be placed across the trough to act as riffles to catch the sand. A convenient size for the trough is 8 by 20 in. inside and 48 ft. long. This is blocked up as high as possible and still permit it to carry the mud from the pipe overflow into the end of a sand settling pit about 45 ft. from the derrick. This pit is made by banking sod 30 in. high around a plot about 10 by 30 ft. From its opposite end an 8 by 10 in. trough (made of 2 by 10 by 16 plank) carries the mud back to the pump pit. This 8 by 10 in. trough is sunk into the ground toward its discharge end and so cuts through the dike around the

[3] I. N. Knapp: Cementing Oil and Gas Wells, *Trans.*, xlviii, pp. 651 to 668 (1914).

pump pit and therefore requires a dike protection. This arrangement prevents surface, tide or ground water from getting into and thinning the mud and was absolutely necessary under the conditions in which the work was done. The wear and tear on the mud pumps is much reduced by thoroughly separating the sand from the mud.

An 11-in. fishtail bit with 6-in. shank and drill collar (about 8 in. O. D.) on one joint of 6-in. pipe and a string of 4-in. drill pipe were used to drill through the 11⅝-in. casing. The same condition of tempering the mud in the surface circulation and in the pump pit before starting to drill should be observed.

Experiences in Drilling.—After the casing was set in one hole at about 700 ft. in clay, drilling was resumed with clear water. The casing shoe did not hold against the difference of pressure between the clear water inside and the thick mud outside of the casing. The heavy mud outside slid down, followed by heaving sand. This made it necessary to pull out the drill pipe to prevent freezing and thereafter to use mud. On running in again, sand was found bridging the hole and it was necessary to rotate and wash out the lower 200 ft. of casing, using a proper mud. The lower end of the casing was thrown out of line so much by caving that the drill-pipe couplings would sometimes catch under the casing shoe. Such experience shows the need of cementing in each string of casing set in unconsolidated formations and the necessity of keeping up at all times a mud of proper temper which is suitable for the work in hand. I found no difficulty in keeping 1,200 ft. of hole open below the first string of casing, with mud, and controlling all water and gas pressure where there was a proven gas pressure of 650 lb. at about 1,600 ft. in depth. I do not know the limit in depth of open hole that can safely be maintained in unconsolidated material when mud is skillfully used. On the other hand, as I have shown, one can get in serious trouble by caving, in drilling only a short distance (less than 100 ft.) below a string of casing when clear water is circulated. It has also been my experience that the circulation of clear water in a well drilled with cable tools in hard ground may bring about serious trouble through erosion and caving. In unconsolidated material the circulation of clear water would mean the probable loss of the well.

I must, therefore, take issue with geologists and petroleum engineers, who recommend washing out wells with clear water before cementing—not only on account of erosion and caving, but on account of the danger of gas blow-outs, and the entrance of oil and ground waters having a deleterious action on the setting of the cement.

I have drilled to 2,007 ft. with the mud circulation showing a temperature of 72° F. The well on being bailed established a flow of 450 gal. of water per minute with a head of 80 ft. and a temperature of 99½° F. This shows that by circulating mud, hot water under 34 lb. surface pressure can be kept back, giving ample time to cement in a casing at ordinary temperature.

Invention of the Use of Mud.—The mixing of clay with circulating water for sealing porous strata in drilling wells is mentioned on p. 167 of *Bulletin No.* 212, *U. S. Geological Survey* (issued in 1903). This is the earliest written mention I find of the use of mud in any kind of drilling.

It was no doubt noted early in drilling by the hydraulic rotary method that after going through a sand or porous stratum and then entering into clay, such porous material was rendered impervious by the clay cuttings; also that caves in sand were built up and plastered over by such cuttings.

The use of mixed mud appears to have had a gradual introduction during the past 20 years and is not the invention of any one in particular. The testing of mud by weighing as set forth in this paper is, I believe, new.

Discussion

A. C. Lane, Tufts College, Mass.—Is there any noticeable effect in drilling against salt water? Mr. Knapp says that the mud settles very quickly when brine is added. What is the effect when you strike a strong salt water in boring?

I. N. Knapp.—You are supposed to have enough mud pressure in your well to keep the salt water out. I have drilled against what I knew to be 80 ft. head of water, and hot water too at 99.5°; this water did not show until the well was bailed and the mud pressure reduced.

Henry Louis, Newcastle-upon-Tyne, England (communication to the Secretary*).—Mr. Knapp may be interested to learn that the use of mud in drilling through porous strata is much older than he appears to be aware of. It has been used notably by Honigmann for boring shafts in Holland and in some of the adjoining portions of Germany. At the Oranien Nassau colliery, shafts about 12 ft. in diameter and 425 ft. deep were sunk in this way—namely, by boring with a rotary borer, a thick mud puddle being pumped into the shaft, so as to maintain a pressure in the shaft greater than the hydrostatic pressure in the surrounding loose sands and thus keeping the sides of the shaft from collapsing. He will find references to this method in *Das Abbohren von Schächten; Handbuch der Ingenieurwissenshaften*, p. 134, also in the *Transactions of the Institution of Mining Engineers*, vol. xiii (1896–97), p. 155. The process appears to have been employed even before 1895, though I cannot give the exact date when it was first used.

A. Beeby Thompson, London, England (communication to the Secretary†).—Mud drilling has attracted world-wide attention, and all engineers associated with petroleum have closely followed its phenomenal progress. As an exponent of drilling Mr. Knapp's writings are known

* Received Jan. 5, 1915. † Received Jan. 28, 1915.

and appreciated on this side of the Atlantic, where we welcome hints and suggestions by practical operators.

I have recently been professionally connected with drilling work of an exceptional kind which has so far defied all efforts to overcome. The human element is too great a factor in drilling to be neglected, but there is reason to believe that several of the drillers engaged were experienced, cautious and thoughtful men; nevertheless they affirm that the troubles are unique and insuperable.

The strata are composed almost exclusively of clay with occasional small lenticles of sand. The clays are believed to be steeply inclined, near the crest of a fold possibly inverted, and the whole mass is permeated with slip planes along which oil occurs. There is no evidence of stratification in the exposed beds, and even surface landslides are frequent along ridges. All the sand lenticles are saturated with oil, and a certain amount of gas is evolved when a depth of a few hundred feet has been reached.

The almost clean clay yields an excellent slimy, viscid mud from which sediment does not settle on standing, and must closely coincide with the requirements laid down by Mr. Knapp. Early efforts with a percussion rig were abandoned as a column of casing could not be kept free any distance, and before 600 ft. was reached the size was too small to go farther.

The rotary was started with 16-in. bit for insertion of 12½-in. drive pipe, and no serious difficulties commenced until a depth of about 500 ft. was reached. Although the thickest mud was employed in circulation, caving or heaving, as the drillers termed it, set in for an hour at a time, and lumps of clay about the size of a hand would be continually ejected, necessitating the employment of several men to clear away the material. Occasionally a block would occur and the pump pressure rose until the obstruction was ejected with force, throwing the mud lumps over the rotary table.

Casing was inserted, but nevertheless only a few feet could be drilled when heaving again commenced, and on one or two occasions the drill rods were twisted off. If the drill rods were withdrawn the hole filled up several hundred feet.

In a series of five wells the following means was taken to endeavor to effect progress: Casing was inserted at intervals and not more than 20 ft. of open hole allowed. This had to be discontinued, as the casing could not be moved after about 400 to 450 ft. had been inserted. Smaller diameters to 6-in. lining tubes were tried to induce more rapid circulation and less area of hole, with no satisfaction.

The casing was itself furnished with a cutting shoe and rotated cautiously while a flush was maintained, but lengths were twisted off by excessive lateral pressure, possibly due to cavings throwing the tube out of vertical.

Finally special heavy flush-jointed tubes were supplied for use either as drill rods or lining tubes. For the former use chilled cast-iron fish-tail bits were screwed to the 8-in. flush-jointed tubes, the object being to smash these up when the desired or maximum depth was attained. As a possible alternative it was intended to drill with 4- or 6-in. drill rods and keep the 8-in. flush-jointed tubes with a shoe near to the bit.

The former project failed, as it was difficult to restart circulation after the brief interval required for adding a new length of tubing, and eventually lateral pressure became so great that the column was twisted in two. The second method was unsuccessful on account of constant caving or heaving which intermittently ensued and prevented progress. Long intervals were allowed to elapse with the pump in operation to remove the cavings, but still progress was impossible.

My personal idea is that the excessive cavings originated from disturbed steeply inclined beds permeated with small slip planes where gas and oil had accumulated. Certainly gas did issue with the circulation mud, proving that the pressure had not been quite overcome by the mud, but the quantity was not such as to lead one to anticipate much difficulty from this cause.

An alternative proposal in view was to use larger pumps and work under a higher pressure, but this entailed an additional outlay not immediately available.

Another cause of the failure of the rotary came under my notice when visiting Colombia last year. Here soft disturbed Tertiary clay shales predominated, and almost endless quantities of material were raised with the mud flush. Eventually mud was observed issuing from the fissures in the ground around the well and operations were ultimately abandoned. Heavy gas pressures were in this case encountered.

Any suggestions from members with experience in mud drilling to deal with problems of the kind mentioned would be a welcome addition to the literature of this comparatively new branch of the oil industry.

I. N. Knapp, Ardmore, Pa. (communication to the Secretary*).—We are under great obligation to Mr. Thompson for the discussion he has presented and the detailed accounts of difficulties met in drilling wells in various localities.

I trust that this will encourage others who have experience in such matters to further discuss the art of drilling, or better, to present papers to the Institute on the solving of well-drilling problems and thus accumulate a record helpful to all interested.

Mr. Thompson does well to call attention first to the human element in drilling. This matter of vital importance is too often overlooked. A driller may be a very satisfactory man in a district he is familiar with,

* Received Feb. 26, 1915.

but this same man when put to drilling under new conditions may be an utter failure and become discouraged and homesick.

The fundamental requirement in all drilling operations to prospect for or produce water, brine, oil, or gas, is to keep the walls of the well in shape so there can be no caving or distortion. Of course it may not always be possible to maintain this ideal condition.

When the walls of a drill hole are self-supporting no special problems are presented, but when such walls require support by using mud very serious problems arise.

The water used in drilling, if not sufficiently laden with mud, may cause erosion and thus start caving of the walls of the wells, or the water may slack the wall material, or the walls may be plastic, and under sufficient pressure will flow into the drill hole. The whole matter may be further complicated by the presence of oil or water associated with gas underpressure. Also nodules, concretions, or fossil wood may deflect the drill.

It needs united effort to solve these problems and I therefore most heartily indorse Mr. Thompson when he says, "Any suggestions from members with experience in mud drilling to deal with problems of the kind mentioned would be a welcome addition to the literature of this comparatively new branch of the oil industry."

The difficulties Mr. Thompson describes remind me of those I met and mistakes I made in the first few wells I drilled with the rotary using mud, although I employed drillers who were successful in other places using this method.

In the first place, I did not appreciate the necessity of keeping a mud circulation up to a standard that would not erode any of the soft sands encountered, and I now know the drillers did not.

Another mistake was that I allowed the free use of the mud circulation when not rotating and with the bit away from the bottom, the idea of the driller being to clear the hole of cavings and cuttings.

We found this was all wrong. When the drilling is skillfully done with a proper mud mixture the drill pipe will run in freely until the bit is on bottom. It should then be pulled back a foot or two until the pump is started.

No mud should be pumped into the well until the pump pit and surface mud are properly tempered. As soon as the well circulation is started the rotation of the drill should also be started. If the bit is left for a long time at any one point in the well without rotating and with a strong mud circulation through it there may be a hole of considerable size washed out from the walls of the well and from which caving is liable to start. Slow rotation of the drill pipe when pumping with the bit off the bottom is necessary to keep the hole cylindrical and stop formation of irregular cavities.

Also, if a drill pipe hangs stationary in a drill hole it undoubtedly lies against the walls in places. Such lines of contact must deflect the circulation to the opposite side of the well and tend to enlarge the hole in that direction and so deform its shape; hence the necessity of some rotation with circulation.

Suppose that in running the drill pipe into a well, in which it was necessary to use mud, the bit meets an obstruction in the open hole 200 ft. from bottom and 300 ft. below a string of casing. If now the pump is started to flush out the well with mud, rotation should be started. If at the same time lumps of clay with sand should appear in the overflow I would consider the mud pump was running too fast and would slow down until the caved material stopped coming. I then would thicken the circulation and try to build up with the mud and stop the caving.

Naturally, the more you flush caved material out of a drill hole in soft ground the more extensive the cave becomes.

Another mistake made was not using a large enough drill coupling back of the rotary bit. At first we used a common 6-in. pipe coupling, 7½ in. outside diameter by 5⅛ in. long, into which to screw the 6-in. shank of a 15-in. bit. In such a case the bit could be deflected 3¾ in. from making a straight hole before the coupling would act as a guide. It is easy to guess that a hard nodule, a concretion, or a chunk of fossil wood, which are common in soft formations, would throw the bit and drill pipe much farther out of line than indicated above and leave a decided hump in the drill hole.

After pulling out and running the bit in again it may hit the hump and cut or knock off the deflecting material and plug the hole. This I believe did happen in my drilling several times, for on rotating to drill out the obstruction probably a new and straight hole would start. We have lost as much as 300 ft. of hole at one time from this probable cause.

A hole drilled with too small a coupling back of the bit is usually more or less humped and crooked and cannot be cased without running in a toothed rotary shoe or a reamer to straighten the hole.

In a 15-in. hole the outside diameter of the drill coupling should be 12 in. by 18 in. in length to be of sufficient size to form a guide to the bit to keep the hole straight and cylindrical and prevent humps.

We made many other mistakes, but they were not as vital to the completion of the hole as those described.

If a hole is not allowed to cave in the least and is kept straight and cylindrical, casing and cementing troubles disappear.

We had twist-offs, collapsed casing, and other troubles, but as we became more skillful, these also practically disappeared.

I cannot concur with Mr. Thompson when he says "My personal idea is that excessive caving originated from disturbed steeply inclined beds permeated with small slip planes where gas and oil had accumulated."

I believe that in any sized hole up to 24 in. in diameter the caving could only have originated from the mistakes in drilling I have outlined. When the cave is once started and gets to be of considerable dimensions, say 4 ft. or more, then Mr. Thompson's explanations would apply.

To illustrate the difficulties that have been overcome in the past, I will quote from I. C. White in the *West Virginia Geological Survey,* vol. i, p. 147 (1899), as follows:

"Soon after the Burning Springs (W. Va.) oil development began in 1860 the petroleum craze spread all over the State. . . . Many wells were drilled in several counties or at least attempts were made to drill them, which nearly always ended by getting the tools fast, and the hole plugged, because the operators had not yet learned the art of dealing successfully with rocks that crumble, or cave, and fall into the hole when water touches them. In the region of Titusville, Oil City and all of northwestern Pennsylvania the rocks (sub-Carboniferous and Catskill) to be drilled through are all hard and the walls of the wells stand firm after the holes are bored, even though drilled 'wet' and full of water from top to bottom. But when the Pennsylvania drillers came down into West Virginia where a much higher and softer series of rocks was encountered (Permian and Coal Measures) and attempted to use the Pennsylvania methods, the result in most cases was failure to sink the borings to any of the Venango County oil-producing sands. Thus it happened that the oil development of West Virginia outside of the Burning Springs and Volcano 'oil breaks' or anticlinal was delayed for 30 years behind that of her sister State on the north."

Cable-tool drilling in both Pennsylvania and West Virginia is carried on by rigs, machinery, and tools of practically the same design for the same size and depth of hole. It is a matter of experience for the driller to use the proper method and manipulation in handling the different ground to get the desired results.

The hydraulic rotary method of drilling with the use of mud when necessary is capable of doing a very much greater range of work than is possible with the cable tools. It is again a question of experience of the driller.

The average driller is very conservative and sticks to what he knows how to do and instinctively tries to use methods he is familiar with to fit new conditions when they may not be at all applicable to the case in hand. He is very resourceful so far as his experience goes.

The oil industry has spread out to such a degree that it has greater need than ever for drillers capable of solving the problems of new fields and their new and variable conditions.

The art of drilling I think offers a great field with ample financial reward to the young engineering graduate who is willing to go into the derrick and work and acquire the "feeling" necessary to really judge of the ground drilled and check the work of the driller.

In many arts it is not at all necessary for the engineer to become an artisan in order to be entirely competent to manage every detail, but drilling does require it.

The Rôle and Fate of the Connate Water in Oil and Gas Sands

BY ROSWELL H. JOHNSON,* M. S., PITTSBURGH, PA.

(New York Meeting, February, 1915)

WHAT becomes of the water which must have filled the oil and gas sands at the time of deposition, has long puzzled students of oil and gas and has found expression in Munn's well-known article on the hydraulic theory. Both Munn and Clapp have pointed out that in the Appalachian field the deeper sands carry increasingly less proportionate amounts of water.

It seems to me probable that the principal source of oil and gas has been in the shales in proximity to the sands which now contain these products. The sand spits which mark off the lagoons from the sea comply particularly with the requirements of a limited sand imbedded in shales rich in organic matter.

Cunningham Craig has objected to this view that the rich organic muds of lagoons are found only at the surface. That such muds do extend to considerable depth I am assured by Prof. Douglas W. Johnson, who has made an extensive study of the Atlantic Coast marshes.

As any particular sand body becomes weighted by a heavier and heavier overburden, the result of increasing deposition, a part of the very large percentage of water in the uncompacted sand and mud is forced out. The water percentage of freshly settled mud is vastly higher than that of the resultant shales, or even of a very porous sand. If beds of less compressible material meet or underlie the shales in question, there will frequently be a lateral motion along such beds to some point where the upward movement will be resumed.

We find in our laboratory work here that oil, gas, and water have extremely slight capacity for gravitational sorting while in a state of rest, but when moving the gravitational sorting is readily accomplished. As the oil, gas, and water pass a body of larger pores, the gas, owing to its lack of capillary attraction, is retained in the large pores.

This has been predicted by Washburne, Blatchley and others on the ground that the water has a higher capillarity and grips the finer pores in such a way that there will be a greater proportion of oil and gas in the large

* Professor of Oil and Gas Production, University of Pittsburgh.

pores. Washburne calls this "capillary concentration." I would prefer the term "selective segregation," because it seems quite possible that other factors contribute as well as differential capillarity. The greater viscosity of oil would cause it to move more readily in larger channels than in small, so that it would with greater difficulty leave the large channels for the smaller ones than would water. Again, even if the viscosity were the same, the walls of the pores are originally water-wet, so that any immiscible fluid such as oil would find penetration into fine pores more difficult than would water and there would be a consequent selective segregation in the larger pores. A fourth factor is the circumstance that the selective segregation of gas in the larger pores would hold some oil associated with it as a pellicle, once oil touched the gas surface, as I have previously pointed out.[1]

We have actually demonstrated in sand arrangements in percolators that the percentage of Texas oil in a coarse-grained body of sand surrounded by a very fine sand becomes excessive where a mixture of oil and water is driven through a water-wet sand. Details of this experiment will be given. A series of experiments is now in progress to distinguish the relative effectiveness of these factors.

The original contents of connate water, its motion caused by compacting, and the selective segregation of oil and gas in the large pores, make it possible to believe that the oil and gas arise in the shale and become segregated in part in the sand, but with much oil and gas remaining in the shale.

David White finds it very desirable to divide the processes of coal formation into two stages, the biochemical and the dynamochemical. He recognizes that methane was formed in the first by the action of bacteria and in the second by the action of pressure. Evidence will be presented to show that we have two periods of natural-gas formation from the organic matter in shales. The first, while the deposit is still shallow and the formation unconsolidated, is effected by the action of anaerobic bacteria. After this action has ceased, a second period of gas formation begins as a result of pressure upon organic matter in the shale. This continues for a very long period. The ascent in lesser quantities of abysmal gas, in part inorganic in origin, as is indicated by the frequent presence of nitrogen, argon, and helium in the deeper sands of Kansas, as shown by Cady and McFarland, contributes also to the high pressure.

A further cause of movement and probably increased pressure with depth, and therefore consequent gas formation (if the hypothesis be correct), is the gradual deposition of cementing material in the sandstone. This reduces the percentage of pore space and displaces material previously occupying such space. The water (with oil and gas, if any) moves

[1] *Science*, new ser., vol. xxxv, No. 899, pp. 458 to 459 (Mar. 22, 1912).

in the direction of least pressure, which is generally upward, as before. But with lowered pressure the water deposits more cementing material, as its solvent ability is thus reduced. The ultimate source of the cement contributed to sandstone is then principally from below. The paths that water takes in its upward course become, by virtue of this fact, clogged, and the difficulties of its circulation are increased with a resulting increase of the pressure gradient.

It has been common to explain the pressure gradient by the "head" of water column constituted by the intercommunicating channels, but the "string of bead"-like arrangements and the tortuosity and fineness of the pores may well make us incredulous of the efficiency of such a head, until experimental demonstration is forthcoming. "Rock pressure" from mere weight of overburden, while generally felt to be inadmissible for moderate depths, can be appealed to at great depths, as well as the expansion with rising temperature of material contained in the pore spaces, as the accumulation of overburden raises the isogeotherms.

We find, then, that the oil and gas reservoirs, when newly compacted, contain a large percentage of the oil which they are to contain. From this time on, there is a gradual formation of cement in the sandstone, which involves a reduction of its percentage of pore space. There is thus forced out of the reservoir some more of its contents; but it will be the water that will be driven out, for reasons given above. The gas which is associated with the oil is gas of the heavy type, such as is usually formed with the oil; but now, owing to the increasing pressure of the continually heavier overburden, we have an additional source of gas formation, namely, the effect of pressure on organic material other than oil which may be in the rock. If we may believe the positive statement of Cunningham Craig, in *Oil Finding*, experiment has actually produced methane in this way. Such a gas would be presumably a "dry" gas.

This recently formed gas, being produced in excess at the points of abundance of the materials from which it was formed, will move through communicating channels in various directions, and will be in large part selectively segregated in the sands. But in so doing it will be a further cause of the displacement of water from these reservoirs.

The expelled water will meet a greater resistance below, owing to the fact that similar processes (only more efficient, because of the greater pressure) will prevent movement in that direction. The amount of possible lateral movement is too limited to offer any important relief. We are forced to believe, therefore, that the expelled water moves, in general, upward, although lenticularity of beds, unconformities, etc., will complicate the course while leaving it essentially a rising one.

Does this not explain why we have, in general, increased gas pressures with increased depth, in view of the fact that direct rock compression and

hydrostatic pressure have been abandoned, for good reasons, as explanations?

Does the adoption of these hypotheses have any bearing on the practice of oil prospecting? It seems to me that, if they are correct, the following practical conclusions result: First, gas is not to be expected in large quantities and under high pressures in relatively recent unconsolidated deposits. Secondly, oil may be expected in good quantities in suitable sands lying in oil shales, no matter how recent, provided only that there has been a proper amount of compacting. Thirdly, a columnar section showing more shale than sandstone, and with its sandstone separated into several beds, rather than constituting one massive bed, may be expected to make available a larger proportion of the oil which was originally formed. If, on the contrary, the view of I. C. White, Thompson, *et al.*, that oil is indigenous to the sands where it is found be accepted, then the ideal section would be one predominantly of sandstone with merely enough shale to act as partitions in facilitating accumulations.

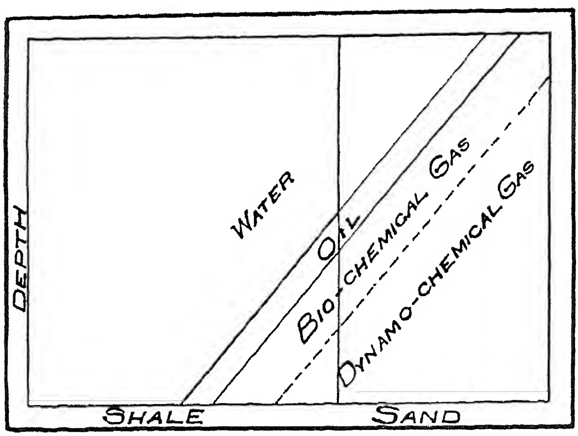

FIG. 1.—DIAGRAM SHOWING RELATIVE POSITIONS OF WATER, OIL, AND GAS.

Fourthly, uniform massive limestones have been rendered so compact by both pressure and crystallization that the oil and gas have been expelled therefrom except in quantities too small to be commercially important. It is highly probable that any porous stratum interbedded with the limestone could receive the oil as it was expelled from the compacting limestone. If we had porous beds more frequently in limestone, we might expect them to be much more important oil-bearing horizons. This view is borne out by the oil formed in the cavities in corals in the Onondaga limestone at Williamsville, N. Y., and yet the absence of oil from the cavities of fossils in many other limestones indicates that some favorable condition is lacking. Is it not possible that in these cases the depth was too great for the necessary bacteria?

With increasing depth, we may expect the reservoirs to show a decreasing proportion of water and increasing proportions of oil and gas, but with the oil not decreasing till a moderate depth is passed, as in Fig. 1; after which there is a decreasing proportion of oil. This assumes that the section continues favorable in essential features for oil and gas. These will vary to such a degree as to make exceptions to the foregoing rule. The gas man may therefore look with more reliance on deep reserves than can the oil man. In the very deep sedimentary horizons, the oil is not absent, if this hypothesis be correct; but is unavailable because, the water having been displaced, the oil has been also in turn displaced from the reservoirs (which are sufficiently porous to be yielding) into the tight grip of the shales.

The freshness of the water in the shallow beds does not make this view untenable because (*a*) relatively little of the deeper waters reach the surface, owing to the interposed barriers; and (*b*) these shallow waters are so much freer in their motion that the slow contributions from below produce little effect upon them.

The view of the ascent of the gases and fluids resulting from a strong pressure gradient from the causes given, rather than a descent of shallow waters to replace a mysteriously lost connate water, has a further application in a consideration of the Gulf Coast oil fields.

It has been supposed that oil was lost at an outcrop because water displaced it by gravity. The theory of ascent leads us to the conclusion that it appears in part in some cases as a result of pressure from below. Fortunately the alternation of coarse and fine rocks through which it passes holds back a good share of the oil. In the Gulf Coast we have an excellent example in the salt-oil-sulphur-gypsum domes. At these vertical passageways we have had not a descent but an ascent, with a consequent formation of the dome. In the Mexican fields the deposit of these substances along intrusives, as well as at faults, are due to the passageways opened by the intrusion, jointing of the intrusive, and attendant disturbance of the adjacent strata, to the ascending current with its load.

Conclusions

It follows from the position here taken that, aside from deviations caused by variations in the favorableness of the accompanying source of hydrocarbons and other modifying influences, we should find an increase of gas with depth, and decreased quantities of oil, as it becomes displaced by the increasing amount of gas, finding its way into the large pores. This is shown diagrammatically in Fig. 1. Should this hypothesis prove correct, the following practical results follow:

1. Much of the world's oil is lost to recovery and there is less hope of deep reserves than we have believed.

2. There are enormous reserves of deep gas, not only in present producing areas but in many others, because, in the absence of water, many more areas without favorable structure will be productive of gas.

3. As the pumps remove the dry gas in these deep pools, the oil back in the shales will release some of its vapors; so that some dry gases will later on, under vacuum, become wet gases and may be profitably compressed, so that the lost oil will not be wholly unrecoverable.

Discussion

A. C. Lane, Tufts College, Mass.—About a year ago Mr. Washburne read a paper on the same characteristic oil-sand waters, discussing their chemical character, and the possibility of their chemical character being modified by adsorption in passing through the shale. We are used to consider circulation as being in one stratum, and I have often gone so far as to think that 2,000 or 3,000 ft. down I could recognize the chemical character as in part connate to a given water. For instance, the St. Peter sandstone, far underneath oil sands, seems to retain a relatively fresh and peculiar character. I am, therefore, tempted to ask the writer if in his experiments he has made any test to see whether the chemical character of these waters is also changed, where there is any adsorption, so that the water passing through these fine-grained beds should leave behind its salt as well as its oil. In many cases salt waters are associated with oil and gas.

A second question has occurred to me. In these days when we hear a good deal about oil flotation, may it not be well to repeat these experiments with pyritic concentrates, and see if they observe precisely the same with regard to the selective accumulation of oil and gas?

Roswell H. Johnson.—I thank Professor Lane for bringing forward this question. I recognize its importance, but am not able to say I have any data upon that point. It is also very interesting in connection with Dr. Day's hypothesis of fractional filtration, and we may very easily get something happening to the water corresponding in some degree to what we get with oil.

D. B. Reger, Morgantown, W. Va.—Mr. Johnson in his paper states his belief that the oil is formed originally from organic matter in the shale, and is forced out of the shale into the porous reservoir of the sand above the shale. He states further, that as you go deeper in the measures where there is an increasing overburden, the oil becomes less and the gas increases in large proportion, or he says that the oil is forced back out of the sand by pressure, into the shale.

Now the oil evidently, in the first place, is forced out of the shale, be-

cause there is a lack of pores in the shale, and is forced into the porous reservoir of the sand.

I ask Mr. Johnson why he believes that the oil in the lower measures, where there is more overburden, is forced out of the porous reservoir, or sand, back into the shale, which we know ordinarily does not contain pores of any kind. I cannot see why it would be forced out of the top of the sand into the shale above, when there is only a little overburden; then, reversely, forced back out of the porous reservoir into the shale again after you have a greater weight of measures deposited above it. His statements are to the effect that oil can be forced out of a porous sand into an impervious shale.

Roswell H. Johnson.—It is unfortunate that in this paper I have been discussing two ideas, and I believe this confusion arises from that fact. I have been talking, on the one hand, of the compacting of the shale, driving its contents along, and then this segregating work which takes place as it passes large pores. Another very different thing is that after the pressure becomes fairly great, a new dynamo-chemical gas begins to form, and this gas, moving into the reservoir, displaces the oil. We are dealing, in the first place, with a purely mechanical feature, a mere pressing out of the fluid contents of the shale, no new gas being formed at all; then later we have this new gas being formed dynamo-chemically, which, going in there, and being held by selective segregation, has to displace the other contents.

I. N. Knapp, Ardmore, Pa.—In regard to the formation of the gas, I have no theory about that, but it is a practical matter that in Louisiana, between 1,600 and 1,700 ft., I got a clear dry gas under a pressure of 650 lb. to the square inch. I went down below that, as deep as to 3,000 ft., and I know of wells being driven to 4,500 ft., in these formations. The fossils saved from wells, up to 2,400 ft. were submitted to Prof. G. D. Harris and Dr. Maury. These fossils indicated to them that we had not reached the Tertiary, that is, that the gas was very recent. An analysis showed that the gas contained 98 to 99 per cent. marsh gas. The drill shows many porous reservoirs that hold oil and gas in paying quantity that are absolutely dry. As for oil and gas traveling through the ground, I have come across conditions where they could not travel through the ground as they were imprisoned in globules in salt and magnetic iron, where there was absolutely no porosity.

Communication to the Secretary*.—After making some laboratory experiments and referring in a rather general way to various theories advanced by others, Professor Johnson offers four tentative hypotheses as having a bearing on the practice of oil prospecting.

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* Received Mar. 3, 1915.

I will attempt to discuss only one of these "practical conclusions," as follows: "First, gas is not to be expected in large quantities and under high pressures in relatively recent unconsolidated deposits."

As the title of the paper relates to a purely theoretical matter and the conclusion quoted relates to practical prospecting only, I am sure I will be pardoned for not discussing the theory.

In looking for oil indications along the Gulf Coast I found there were very many gas escapements in Terrebonne Parish, La. A few were found highly impregnated with sulphuretted hydrogen. I had numerous samples taken for analysis and the result seemed to indicate that oil might be found by drilling.

Fig. 2 shows a typical gas escapement in a field.

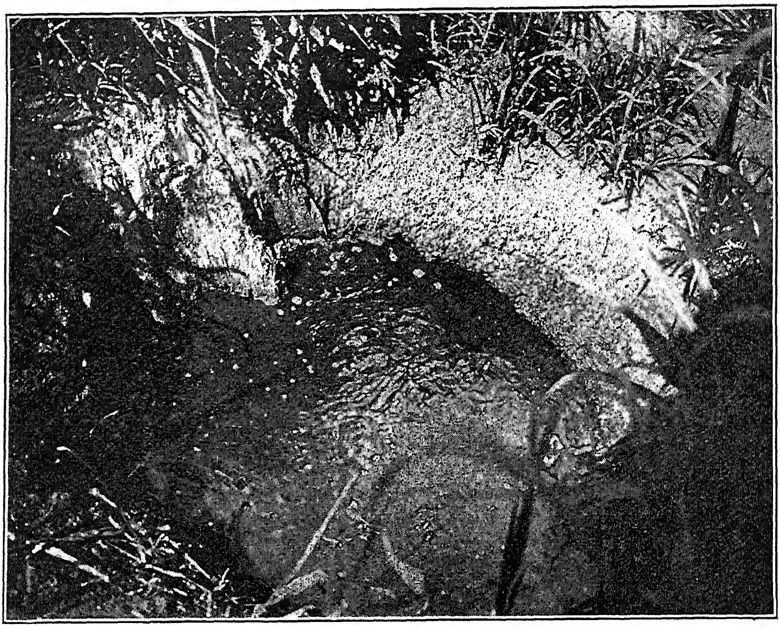

FIG. 2.—GAS ESCAPEMENT IN A FIELD, TERREBONNE PARISH, LA.

These escapements are fairly constant in volume and have been known for many years.

The discovery of deep gas in the Parish mentioned was made on the Lirette farm in Sec. 51, T. 19-S, R. 19-E. (see *Bulletin* 429, *U. S. Geological Survey*, p. 163).

This well blew out from 1,710 ft. and continued to blow for 99 days at an estimated rate of 5,000,000 cu. ft. per 24 hr. without any apparent weakening, and then it caved in and shut off. The first 24 hr. it made dry clean gas and then began to spray salt water.

Previous to the blowout of the Lirette well and about 5 miles from it,

I drilled a prospect well over 2,400 ft. deep in unconsolidated material. I submitted to Prof. G. D. Harris and Doctor Maury fossil shells from this and other Terrebonne Parish wells (see *Bulletin No.* 429, *U. S. Geological Survey*, pp. 169 to 173). They definitely state "that none of these wells have penetrated beds that can be regarded as older than Pleistocene." I presume therefore it will not be questioned but that these are, geologically speaking, "relatively recent unconsolidated deposits."

After the Lirette well blowout I drilled four wells within about a mile of it in an effort to make a commercial development. Gas in quantity and under great pressure was found in each well from the same probable horizon.

Below 1,500 ft. thin layers of slightly cemented sands were found that were probably true gas sands and these were bedded with thin layers of loose sand, black clays or shales, and shell conglomerates. There were salt-water sands above and below near the gas, but I was unable to locate them definitely. Below 700 ft. the clays were very dry and these dry clays below 1,200 ft. would flow when the pressure was taken off by bailing a well and gradually close it up.

Two of the four wells mentioned were lost by blowouts that lasted about 24 hr. before the wells caved and shut off. I feel confident that each of these wells was making gas at a rate exceeding 20,000,000 cu. ft. per 24 hr. for some hours before it shut off. I base this statement of quantity on observations made previously on many wells of measured capacity.

I saved one well as a gas well and it gave dry clean gas for a couple of months, when it was shut off by clay squeezing in. This was in the 1,600 to 1,650 ft. horizon. The closed pressure was 650 lb. and the initial open flow capacity 2,000,000 cu. ft. per 24 hr. An analysis showed 98 per cent. marsh gas and no illuminates, which I think indicates the entire absence of oil. There was not the slightest taste or odor of petroleum.

I believe I have shown by the drill that gas may exist "in large quantities and under high pressure in relatively recent unconsolidated deposits" and this is diametrically opposite to the author's first conclusion.

There have been numerous gas blowouts from recent deposits in the Louisiana and Texas oil fields contiguous to and over productive oil pools. I witnessed several such blowouts from comparatively shallow depths and their violence indicated the gas was under high pressure.

I can refer to many published accounts supporting my showing. For instance, in *Petroleum Mining*, A. Beeby Thompson, under "Evolution of Gas" (pp. 94 to 100) cites many instances of large volumes of gas escaping from unconsolidated material in the various oil fields of Russia, Burma, Borneo, Peru, and Trinidad.

One view (Plate XIII) shows where "Many thousand tons of puddled clay were ejected in a few hours, accompanied by the evolution of hundreds of thousands of cubic feet of inflammable gas."

The *New Orleans Times-Democrat* of Nov. 26, 1911, gives an account of a gas eruption in the sea off Erin Point, Trinidad, on Nov. 11, 1911, and says "The next day it was found that an island of 2½ acres had been formed. A landing party found the place still warm, and by laying down boards were able to examine two cones, 12 to 15 ft. high, from which gas was escaping. The air was saturated with the odor of sulphur and oil."

An observer of this eruption told me the gas made a flame over a

FIG. 3.—TYPICAL MUD CONES FORMED BY GAS, TRINIDAD, BRITISH WEST INDIES.

mile high for perhaps half an hour. His estimate of height he checked by a rough triangulation.

Fig. 3 shows typical mud cones of medium dimensions such as form in the oil and pitch lake regions of Trinidad and Venezuela.

Natural gas is continually forcing up cones of pitch in the pitch lakes and away from them cones of clay or mud, and they break at the apex and gas issues that may be lighted and sometimes burns for days. Occasionally pitch and oil show with the gas. I have this information from a man who lived for some months in this region and drilled some deep wells in Trinidad. He also said the gas found below the surface increased with the depth and pressures of between 400 and 500 lb. were expected around 1,200 ft. in depth.

Peckham reports on the pitch lake of Trinidad as follows: "As the bitumen rises in the center of the so-called lake it is inflated with

gas . . . . the gas cavities are of all sizes, some of them very large, and in the aggregate occupy, at a rough estimate, from one-third to one-half the volume of the pitch." (See *American Journal of Science*, 4th ser., vol. i, p. 193.)

Mud lumps are common in the delta of the Mississippi. The mud slowly accumulates around the gas and sludge vents and forms cones ranging in height from a few inches to several feet. These lumps are largely composed of structureless clay. (See *Professional Paper No. 85-B, U. S. Geological Survey.*)

It is very likely the gas would form cones in Terrebonne Parish under proper conditions, but the drill shows numerous loose sand strata and shell beds capable of relieving in a lateral direction any tendency of a rise in gas pressure sufficient to form cones.

I am sure my references also show that gas may be expected in large quantities in relatively recent unconsolidated deposits.

E. W. Shaw, Washington, D. C. (communication to the Secretary*). —With the word petrology—literally the science of rocks—being used primarily for igneous rocks, and with geophysics coming to mean the study of lavas and the rocks into which they solidify, contributions to the knowledge of the physical chemistry of sedimentary rocks, even though only ideas which may be used as working hypotheses, become most welcome. The forces which control the formation and subsequent history of the stratified portions of the earth's crust are so little known that no one has yet been able to settle finally the questions as to whether petroleum is of organic or inorganic origin; whether it originated far from its present position, or nearby. Yet both the chemical and physical problems of the rocks in which oil occurs seem to be more simple than those of the igneous rocks. The oil-bearing rocks of western Pennsylvania, for example, have in all probability never been subjected to a pressure exceeding 3,000 or 4,000 lb. to the square inch, nor to a temperature higher than 100° or 200° C. These pressures are considerably below the crushing strength of rocks, and the temperature far below their melting point. The igneous rocks, on the other hand, have been affected by enormous pressures and very high temperatures and yet steady progress is being made in reproducing in the laboratory the conditions of their formation and subsequent modification within the earth. The solution of the sedimentary-rock problems would seem to be fully as important as those of igneous rocks, and yet we must content ourselves with reasoning backward and our inferences as to what has happened are oftentimes little more than guesses.[2]

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* Received Apr. 16, 1915. Published by permission of the Director, U. S. Geological Survey.

[2] The need of work on sedimentary rocks is indicated by a question put to the writer by an eminent authority on economic geology, to this effect: "Is salt water at all common in oil fields?"

One of the fundamental problems is the origin of the salt water in oil fields. Is the salt water fossil sea water; has it been formed by the solution of salts of various kinds from distinct beds, from disseminated particles, and from the weathering of suitable grains in the rock; or is it of deep-seated origin? Its general resemblance to sea water points to the first, but the principles held by some concerning the free circulation of water throughout the earth's crust, and certain peculiarities of composition indicated by the few analyses available, seem out of harmony with such an explanation.

We must grant that the strata before becoming deeply buried, if not at the exact time of their formation, were saturated with water. Those containing the remains of marine organisms or interstratified with such beds were, except for gas bubbles, filled with sea water. Many layers of sand which were once parts of sand dunes, and also many layers of sand deposited in fresh water, have later been submerged beneath the sea or lowered to such a position that salt water flowed into them. Some beds of fresh-water origin have never been below sea level, and hence may never have contained any salt water, but such beds form only a very small fraction of those in any way associated with oil and gas.

It is also incontrovertible that the sedimentary strata are now more compact than they once were, and that part of the water with which their pores were filled at or shortly after their formation, has been forced out; also that shale has suffered much more compacting than sandstone. It seems reasonable to assume, further, that the main part of the compacting took place in each stratum within a comparatively short time after its formation, and that probably the time of greatest settling was shorter for sands than for clays, and probably shorter for clays than for coal. But beyond this, inferences concerning the history of connate water are little more than assumptions. Has it remained in the rocks since their formation? It is unthinkable that each pore has retained its own water from the start. Then how far has it traveled: to other parts of the same bed, to other beds in the region, or to other regions?

In the first paragraph of his paper Professor Johnson uses the expression, "The deeper sands carry increasingly less proportionate amounts of water." This conclusion has been frequently stated, and seems to have gone unchallenged, but the writer has searched in vain for prooftht a the deeper sands actually do contain less water. The idea seems to be held that certain deep-lying sands have empty pores, as is shown by the fact that they absorb water instead of supplying it. But cannot all the phenomena be explained by the factors which control the flow of water in deeply buried porous strata? Is it reasonable to suppose that the pores of any sand have nothing in them? If they contain water, oil, natural gas, or air, they cannot absorb more liquid or gas unless some of that which they contain is either given up or compressed into a smaller space. In

general, the deeper the sand below the surface, the more likely it is to be effectually sealed in; hence it is conceivable that a deep sand saturated with water may be so completely surrounded with shale that no air or other gas or liquid can enter it to take the place of the water which would otherwise flow out into a well. Furthermore, it is also conceivable that a sand at any depth connected with the surface at, for example, two widely separated points, may have water flowing through it under such a slight head that the weight of the water in a well which is sunk to the sand may cause it to flow from the well into the sand. The sand may thus have its pores completely filled and yet actually absorb water. If the known facts allow of such an interpretation, the reports of deep-lying dry sands, or even of less water in deep sands, should not be taken as final.

In the second paragraph, the belief is expressed that oil and gas have originated in shale near their present location. The field evidence seems to the writer to admit of quite a different interpretation, though neither interpretation is sufficiently well supported to be more than an idea or working hypothesis. The writer has found that recent mud deposits along the sea coast have a weight per cubic foot ranging from a little more than that of water to about twice that of water. The total amount of solid matter per cubic foot in some samples weighs less than one-tenth as much as a cubic foot of the same matter without pores of water would weigh. The water content of various muds tested ranges from 40 or 50 per cent. to 90 per cent. or more. In the compacting of such muds into shale, a large part of this water is expelled, and if oil and gas originate in such muds, the possible distance to which they may be carried by such outflowing water seems to be great, for not only a part of the water originally contained in a particular part of a bed moves through it, but also water from other beds and other parts of the same bed.

The idea that sand spits become buried and some of them form oil reservoirs is commonly entertained, but seems to the present writer quite untenable. The material of sand spits, beaches, etc., is rarely buried without being completely reworked, and changed into some other form of deposit. In an advancing sea beach deposits are formed at the shore and gradually pushed inland as the sea advances. They are continually subjected to a reworking process, and such parts as are not carried landward by the advancing sea are spread out in the form of bottom deposits.

That organic muds lie at various positions below the surface, and in places are continuous to great depths, seems too well established to need argument. At the mouth of the Mississippi, for example, carbonaceous muds from which great volumes of marsh gas are given forth are found at all depths down to nearly 2,000 ft., the depth of the deepest well yet bored. The argument of Craig referred to seems somewhat misleading, but not altogether incorrect. Craig says that "The fallacy lies in the assumption that these samples from the upper layers of sludge (in harbors, mud flats,

etc.) are typical in chemical composition of the mass of slowly accumulating material beneath." Studies made by the writer indicate that more organic material is present at the bottom than a few feet below, but it is nevertheless found at all depths. Craig argues that all or nearly all animal matter oxidizes before it is buried, and the writer knows of no evidence in recent deposits which is incompatible with this view. The buried carbonaceous matter may be almost entirely vegetable, but at the same time, Professor Johnson's belief that organic matter is present at considerable depth is well supported.

The idea is conveyed that the water which is being pressed out of layers of mud and sand takes a course which is dominantly upward. But oil-field strata are commonly nearly horizontal, and hence, water which is being pressed out of mud and sand should move much more laterally than vertically. Also, in order to have an outlet at the surface, the sand need not be a single persistent bed extending to some point sufficiently low to allow of flow in that direction, but it may have zigzag connection, through interfingering lenses of sand, with the surface at some more or less distant point.

The possibilities of gravitational sorting depend upon several factors. To some investigators of petroleum it has seemed quite possible that without any lateral movement, oil, water, and gas may take positions with reference to their specific gravities within a layer of sand, the particles of one fluid moving past another in the minute pores of the sand. A few years ago the writer obtained some glass tubes of various diameters, up to about ¼ in., and 6 ft. long, and filled them with alternating globules of oil, gas, and water, in order to determine how large a pore is necessary for gravitational sorting. M. J. Munn, whose hydraulic theory was such an important contribution, was associated in the experiment. It was found that only in the larger tubes was there any perceptible movement. Some would probably remark concerning such experiments, that what would not happen in the laboratory might happen in nature, because of the millions of years available. This suggestion seems inapplicable, because in a tube more than 50 times the diameter of the average space between sand grains in oil reservoirs no movement took place in several weeks. The frictional resistance to movement increases so rapidly with decrease in size of pores, that it seems beyond possibility that oil and water could change places under the influence of gravity in such small spaces as are available in ordinary sands.

The only possibilities of gravitational sorting without other movement appear to be the following: First, in a coarse sand with very fine globules of oil in water—almost an emulsion—such a movement may take place; second, if through surface tension, slight movement, or some other factors, oil comes to fill the pores in a considerable body of sand in the lower part of a stratum elsewhere saturated with water, it seems possible that

the body of oil as a whole might be floated up to the top of the sand and water take its place below, without any lateral movement. If, however, movement can be postulated, gravitational sorting may take place very readily. To illustrate, if a pore between sand grains contains both oil and water, and there are two openings leading to adjoining interstices, it is easy to imagine that the oil would take the upper and the water the lower opening, other conditions, of course, being well balanced.

The expression "capillary attraction" seems to be frequently used in literature on petroleum without an adequate realization of its controlling factors. As a result, conclusions are sometimes drawn and statements made which are open to question. For example, in Washburne's important contribution on Capillary Concentration,[3] it seems to be assumed that capillary attraction is independent of the material forming the sides of the capillary opening. It is stated that "liquids are drawn into these pores by the force of capillary action, which varies directly as the surface tension of the liquid and inversely as the diameter of the pore. Crude oils probably have only about one-third the surface tension of water, and hence only about one-third of the capillary power." Now, the fact is that capillarity depends not only on the surface tension of the liquid but on the nature of the surrounding substance. It is, hence, unsafe to conclude from determinations of the height to which water rises in a glass tube that a similar force will be exerted upon water by clay or some other substance. A glass tube in which water will rise to a height of an inch or a foot will, if greased, have quite a different effect upon the water. Indeed, its pull may be negative, so that the water in the tube may stand lower than in the surrounding vessel. It must be remembered that capillarity requires three substances, water, glass, and air, for example, and that at the contact between shale and sandstone saturated with water, no capillary attraction will be displayed.

As to the relative viscosity of oil and water, it seems probable that the difference, particularly in deeply buried sands, is not important, because the temperature is higher and the viscosity lower than at the surface. The difference in viscosity between warm oil and cold oil is greater than that between average oil and water at ordinary temperatures.

The statement to the effect that oil does not readily penetrate pores whose walls are wet with water seems at first thought reasonable. We oil our boots and our roads partly to make them shed water. But it may be remarked that both water and oil are attracted by the clay and sand walls of pores, and the frictional resistance to movement of the fluids thus gripped is great, at least for the molecules adjacent to the walls. On the other hand, water does not show such an attraction for oil, and hence a minute globule of oil may pass quite readily through a pore having a

[3] *Trans.*, l, p. 830 (1914).

larger diameter lined with water. Globules of oil inclosed in water in a glass tube rise more readily to the top than oil globules which come into contact with the glass. However, one may reasonably doubt the existence of oil globules sufficiently small to pass through the finer pores without touching the walls.

The amount of oil which may be carried by gas bubbles as films lining the bubbles is apparently small, for the oil films are little if any thicker than molecules and a very large amount of gas would be required for the transportation of a minute quantity of oil.

Professor Johnson accepts the idea that some oil and gas are produced by pressure. From the standpoint of physical chemistry this seems somewhat doubtful, and poorly supported by laboratory results. The few experiments that have been performed seem to indicate that when carbonaceous deposits are subjected to great pressure in the laboratory, they are not metamorphosed, and no oil, and commonly no gas, is produced. If water is present and allowed to escape the sediment is compacted, but if the water is not allowed to escape the sediment is unchanged. The conclusion that oil may be produced by dynamochemical agencies appears to rest upon rather inconclusive geological evidence, and to be unsupported by general principles. On the other hand, if the temperature is raised, it is not at all difficult to obtain various hydrocarbons from any carbonaceous deposit. A little more geophysical investigation of this subject is greatly needed.

The inference that some oil is biochemical is also in need of better support. If it is correct, oil films should be common at the surface of recent deposits, but they are very scarce. In fact, the writer has not found them except where some animal remains were decomposing. Bacteria do not thrive far below the surface. They have been reported at 20 ft., but this report is doubted by some. The writer has sunk many holes in marshes, to depths of 50 to 100 ft., but has found no sign of oil.

The idea that the character of coal and oil of western Pennsylvania shows a relation to the horizontal thrusts which have produced the folding in the Appalachian Mountains—that the thrust pressures have affected the rocks west of the Alleghany front in a gradually decreasing amount, seems to the writer rather improbable. About the only indication of thrust in western Pennsylvania and Ohio is the fact that the anticlines and synclines have a general trend more or less nearly parallel to the Appalachians, but can such low and multitudinous folds be produced directly by lateral thrust? Where beds first begin to yield to lateral thrust they are thereby weakened and there is a tendency for the shortening to be effected by a single high fold, though this tendency is modified somewhat by heavy loading. The shortening involved in the low folding of the oil and gas region is so slight, and the wrinkles are so generally developed, that it seems much more probable that they were produced by forces act-

ing principally from below, among which gravity is dominant. An anticline 30 ft. high and 3 miles broad means a shortening of only a tenth of a foot. According to Bailey Willis, in the eastern part of the Appalachian region the principal component of the thrust was horizontal, whereas in the western part it was vertical. If the quality of coal and oil west of the area of pronounced folds shows a relation to the Appalachian Mountains, may it not be due to greater loading on the east? Some of the formations are known to thicken in that direction, and there is good evidence that more material has been removed by erosion from eastern Pennsylvania than from the western part of the State.

One of the most interesting parts of Professor Johnson's paper is the paragraph showing that he regards it possible that some gas is of inorganic origin. The close confinement of most gas pools makes the possibility of accession of deep-seated gas seem remote. The fact that gas will be held under a pressure of hundreds of pounds to the square inch for many years by a stratum of shale a very few feet in thickness, makes it difficult to believe that any gas has traveled upward through many thousand feet of very compact rock; yet the state of knowledge of sedimentary rocks is such that the conclusion that such a migration is impossible would be altogether unwarranted. Besides, there is the possibility of transfer by solution, for water will dissolve hydrocarbons to some extent.

The conclusion is drawn that the cement in sandstone comes principally from below. Is it not quite possible, however, that a large part of the cement came from within a small fraction of an inch of where it is now found? Granular materials under pressure, containing water under less pressure, tend to dissolve at the points where the grains are in contact, and the dissolved material is commonly redeposited very near the point of solution. If the water and rock are affected by the same pressure, solution and redeposition will not take place, and this may explain the varying degrees of cementation displayed by sandstones of apparently similar history. Even where the cement is not of the same material as the sand grains, is it not possible that it has come from various directions, according to the movements of the water?

In connection with the idea that temperature rises in great masses of sediments on account of rise of isogeotherms—that these masses have sunk into the earth to positions where the temperature is higher, it should be remembered also that on account of deformation and erosion, the temperature throughout the mass of strata has been affected at different times in different directions. It has seemed to the writer that these changes may play a part in the accumulation of oil and gas and water, because any little motion, if it amounts to a distance of only a few times the diameter of a pore, and especially if oscillatory, aids in the segregation of oil and gas.

The inference that, other things equal, the greater the pressure the

greater the amount of gas, and also that high pressures tend to produce gas rather than oil, is, if true, of great importance, but at present it seems to be in need of more supporting data.

The point is made that tortuosity and fineness of pores affect the head of water. Is hydrostatic equilibrium affected by such factors? Where a fluid is moving through pores in a rock, hydraulic forces are involved and the effect on the water head may be considerable, but where movement is wanting, the fluid should in time come to equilibrium and the head of the water should be the same without regard to size or tortuosity of the pores unless gas or oil is mixed with the water so that capillarity or some other factor enters the problem.

The height to which water will rise in a well may be affected also by atmospheric pressure if the sand is an effectively sealed-in lens. The areas of the upper and lower surfaces of a sand layer are generally so vast, compared with the areas exposed in the sides of a well, that as a general rule there is, somewhere, a few acres where water may slowly enter the sand over an area so much larger than the walls of the well that it may take the place of a considerable volume which flows out into the well. Since the greater the depth the greater the possibility of sealed-in sands, the lower sands should in general yield less water and be reported drier.

The head of water in a sand evidently depends on several factors and has a wide range. In a sand outcropping down the dip but not up the dip, it may be zero or even a minus quantity, so that water will flow from the well into the sand. The maximum height to which pressure may rise is even more problematical. Obviously it may range up to the weight of a column of water, or nearly 450 lb. per square inch for each thousand feet of depth. In lenses which become effectually sealed in before settling ceases, the pressure may become as great as the weight of the superincumbent load of rock, or about 1,200 lb. for each thousand feet of depth. If in such a sealed-in lens some chemical change takes place, such as the development of gas or precipitation of cement, it seems conceivable that the pressure may, through the tensile strength of the rock, rise even higher without producing an upheaval, just as the pressure in a boiler may be greater than the weight of the steel forming the upper part of the boiler. If such pressure were developed over a very large area the coherence of the rock would of course become of small importance, but oil and gas pools are not so large, and the pressure does not rise so high as to make the coherence of the hundreds of feet of strata negligible.

The conclusion "We find, then, that the oil and gas reservoirs, when newly compacted, contain a large percentage of the oil which they are to contain," does not seem to be in harmony with the character of recent deposits. The Mississippi delta, for example, contains a great abundance of organic material of vegetable, and perhaps some of animal, origin, and from this material immense quantities of marsh gas develop,

but no trace of oil has been found, though several wells have been sunk to depths exceeding 1,000 ft. Further, no hydrocarbon gas, other than methane or marsh gas, has been found.

The lateral movement of water and oil is thought to be slight (p. 589). But in upward movement the fluid must traverse clay and shale in which such movement is opposed by not only hydrostatic head but great friction. On the other hand, in a direction more nearly horizontal the fluid may flow through sandstone where the hydrostatic head is nearly the same and the frictional resistance to slow movement is probably much less, though the distance is greater. The facts that oil and gas generally occur in great basins, that sands may either extend out to the surface or have a connection somewhere with some other sand which outcrops toward the margin of the basin, and that except in hard fractured strata it is easier for fluids to move along beds than across them, make it seem probable that the fluids in being forced upward move much more laterally than vertically.

Concerning the statement that "gas is not to be expected in large quantities and under high pressure" in recent deposits, it should be remarked that immense quantities of marsh gas are included in recent deposits, though ordinarily not well segregated, so that, on account of dissemination and the difficulty of recovering gas from soft materials, few attempts are made to collect and make use of it. May not the greater amount of gas recoverable from deposits which have become somewhat consolidated be more reasonably ascribed to the fact that there has been sufficient time for accumulation in well-defined pools and the clays have become sufficiently tight to retain the gas, rather than to a greater production of gas at a later stage on account of greater pressure?

Would not the third conclusion (p. 590) be more correctly stated if the wording in the latter part were "make available a larger amount of oil" instead of "a larger proportion of the oil which was originally formed?" The proportion of available oil to organic material might, it would seem, be fully as great in an area of thin shale and thick sandstone as in an area of thick shale and thin sandstone, particularly since sandstones are sufficiently variable in porosity to affect the location of pools. Whether Johnson's conclusions or those of the other investigators referred to are correct, it may be contended that the oil in thick sands has a better chance to escape than that in thin sandstones.

Professor Johnson evidently regards as a possibility, the idea that a considerable amount of oil may have come from limestones (p. 590). This inference seems to be opposed by certain facts, one of which is that calcareous and carbonaceous materials are rarely deposited together. In peat bogs shells are generally absent, and in seas where lime is being deposited there is generally complete decay of the organic matter. Beds of limestone, coal, and carbonaceous shale are commonly interstratified, but the materials are generally not mingled in a single layer. The char-

acter of the very recent lime deposits in Florida, for example, is such as to make one dubious concerning the conclusion that a globule of oil in a Paleozoic coral is the remainder of a part of the body of the coral. It would seem much more likely that the globule of oil had migrated from some other place and stopped in the coral because of the cavity. How far the globule may have traveled one may scarcely guess. Some phenomena displayed by oil suggest that, given geologic time, it may journey for very long distances. It has been spoken of as a "wandering Jew" the possible extent of whose travels is almost unlimited. On the other hand, the effective inclosure of oil and gas pools and the high pressures developed tend to make one assume that the oil and gas have not migrated far. Perhaps, however, the effective sealing-in of pools may take place long after the deformation of the stratum, and may be preceded by a long time, during which migration is easy.

Concerning the depth and conditions under which bacteria may live and operate, knowledge is scant. Anaerobic bacteria have been reported at a depth of 20 ft. in bogs, but whether they are sufficiently common at such depths to be important factors, to say nothing of whether they may effect important changes at greater depths, remains to be determined.

The conclusion that gas is more abundant than oil at great depths requires that pressure is more effective in producing gas than oil, and that neither gas nor oil travels far. The idea that the oil is unavailable in deeply buried strata because it has been forced into the shale, seems out of accord with the nature of strata which have been deeply buried. There is much pore space in Cambrian sedimentary rocks, and therefore for oil to be forced out of them requires that some of their pores be filled with gas and that no openings out of the surface exist. On the contrary, Cambrian sandstones are commonly loose and porous, and the writer would expect that even if they were at first filled with oil, the pressure would push the oil diagonally along the bed toward the outcrop before it rose high enough to force the oil into shale. If the sand were not saturated with oil at the start, the remaining pore space, except near the outcrop above the surface of ground water, would be occupied with water which would all be forced out, either into shales or at the outcrop, before any oil would enter shale.

The statement is made that "relatively little of the deeper waters reach the surface, owing to the interposed barriers." If this were true the waters of the deeply buried sands should be under greater pressure than that exerted by the superincumbent load of rock, and when drilling deep sands containing water only, we should have a series of water gushers each followed by a certain amount of settling. On the contrary, it seems to the writer probable that the water in all sands is gradually forced diagonally upward to the surface through settling of strata and filling of pores, and that, although great resistance is sometimes encountered, this resistance,

except in sealed-in lenses, rarely amounts to more than that of a hydrostatic head of water, filling the overlying rocks.

Is it true that the opinion is generally held that oil is "lost at an outcrop, because water displaces it by gravity"? The ideas that some seeps are produced by oil being forced out by pressure from below, or that some are due to hydraulic pressure exerted through the stratum from which they arise, seem tenable. Whether in the Mexican fields the oil adjacent to intrusives has been carried up by water squeezed out of the strata below, or has been carried by the intruded mass, or has not been carried up at all, but has formed in approximately its present position, are unsettled questions.

The conclusions at the close of the paper are based upon the preceding argument, and insofar as it is uncertain, the conclusions are doubtful. The second conclusion, in particular, seems to the writer to need modification, because it involves "absence of water" in deep sands. This inference, as stated before, though generally regarded as well established, seems very unlikely on theoretical grounds, and not proved by field evidence.

Professor Johnson's inferences are evidently the result of much careful thought and although the present writer feels inclined to a different opinion concerning many of them, few can now be actually disproved. In any case, the paper is full of ideas and stimulating to thought.

C. W. Washburne, New York, N. Y. (communication to the Secretary*).—Mr. Johnson recognizes that connate waters must be driven outward toward the earth's surface. He considers the ascent due to the settling and packing of the underlying sediments and to the formation of gas within the rocks, supplemented by abyssal pressures.

The origin of the methane in the sedimentary strata probably is not simple. There are three possible modes of origin: (1) as marsh gas formed near the surface and entrapped in the sediments; (2) from abyssal sources; (3) by the decomposition of organic matter buried in the sediments. The abyssal source is generally negligible, and in all cases doubtful. The third origin is the most important in the gas fields. A fourth mode of origin, by the dissociation of liquid hydrocarbons, frequently has been suggested, but it is improbable, because crude oils are exothermic. Therefore they have no tendency to split or decompose at low temperatures, as do endothermic substances, such as coal, which is altered geologcally by the loss of methane, water, and carbon dioxide. This cannot happen to a hydrocarbon until its temperature of dissociation ("cracking point") is reached, which is over 250° C. for the most easily cracked hydrocarbons and over 400° C. for others.

Engler says that the dissociation of fats and waxes into oils is an endothermic process, thereby unconsciously nullifying his own theory of the

* Received July 6, 1915.

dissociation of these substances at low temperatures. However, recently I have found reason to think that heat is liberated by the disruption of the molecules of fats and waxes, as it is in the molecular decomposition of the substances of wood, coal, etc. If this tentative opinion is correct, and Engler wrong, then it seems that the Engler-Höfer hypothesis of the origin of oil becomes not only possible, but even probable.

A second conclusion is that all complex organic molecules which remain in their original state to the extent of retaining combined oxygen must have a tendency to decompose, with the production of exothermic substances, including methane and other hydrocarbons. This would result in pressure, since the products occupy greater volume than the parent substances. To reverse the problem, as in the paper, and to say that pressure itself would cause the production of methane from organic matter, is a confusion of cause and effect, and seems to be contrary to physical principles. Those who appeal to pressure in this way probably are overlooking the essential factors. Since the investigations of Brame and Cowan[4] and of others have shown that coal is endothermic, its natural underground alteration, even at low temperatures, is explained readily by the common principles of thermo-chemistry. The expression "dynamo-chemical stage" of alteration could be replaced advantageously by the better established word, "thermo-chemical."

Abandoning pressure as a cause of the liberation of methane from organic matter does not invalidate Johnson's main conclusion that the formation of methane from entombed organic matter probably increases with depth, because a downward increase would result from the increase of temperature with depth, and also from the greater age of the lower strata, in which the process has been longer in operation.

Naturally I agree with Johnson that the formation of methane crowds away an equivalent volume of water, which must move upward toward the surface because there is no other direction in which escape is possible. The water squeezed out by the settling, consolidation, and cementation of the sediments also must escape upward. Locally also there is reason to believe in the ascent of abyssal fluids, and the pressure of the latter probably supplements that arising from the formation of methane and other gases. I had independently reached the conclusion that there is a general ascent of rock fluids across the strata, from a study of the general distribution of gas pressures, and from the abnormally high chlorine content of oil-field waters.

The evidence of this ascent and the reasons for believing that it is a process in general operation cast doubt on the existence of true connate water in any stratum. The connate water would be shoved out or modified by the entry of other water from below, and rarely could any water

[4] *Journal of the Society of Chemical Industry*, vol. xxii, No. 22, p. 1230 (Nov. 30, 1903).

strictly be connate to the strata in which it is found. Originally it may have been connate with some lower strata, which in the Appalachian fields would be in early or pre-Paleozoic formations, which were laid down at a time when the sea was comparatively young, and when theoretically the sea-water ratio of Cl:Na may have been as high as that in the waters of deep wells. To some geologists this explanation will be more acceptable than the alternative, that juvenile water rich in calcium and magnesium chloride has entered generally into the strata of oil fields and in many other places; but whatever explanation of this is preferred, the conclusion remains inevitable from any point of view, that water and probably other rock fluids have risen far across the strata. Probably there is no free water left in the older strata that strictly is connate with these formations.

In the paper on the capillary concentration of gas and oil, I avoided reference to the possible influence of viscosity in promoting the concentration of oil in the larger pores, because it would operate against the apparently identical segregation of gas. Moreover, I have come to think that most of the migration of oil in fine rock pores takes place when the oil is still comparatively light. (I hope soon to present the geological and chemical reasons for believing that all crude oils are descended from lighter parent oils.) At the temperatures prevailing at depths of 2,000 m. light oils have approximately the same viscosity as water, some even less, so that viscosity cannot be an important factor in the concentration or segregation of such oil in its original migration. In the later and comparatively shallow migrations of the condensed heavier oils, such as a typical fuel oil of California, viscosity would be a determining factor in causing the oil from fissures to enter sandstone rather than shale. In fact, such oil could not enter even dry shale, and no oil could enter wet shale from fissures if the conclusions from capillarity are correct. However, fissure migration is a secondary feature, although of great local importance in the formation of some "pools." The original concentration of the oil in sands and fissures must have been due to capillary action operating on comparatively light oils. There is no other way by which the oil of the California fields could have been removed so nearly completely from the diatomaceous and foraminiferal shales in which it is thought to have originated. If the oil were then in its present viscous, highly asphaltic condition, only a very small part of the oil could have been removed and the shales would be generally petroliferous and dark colored, instead of being very locally saturated and generally light colored, as they actually are. In other words, I accept Arnold's conclusion that the oil of California gets heavier as it migrates, and I believe also during its storage in the sands, which is in accord with all sound observation and theory, and I oppose the view of Höfer that crude oil grows lighter with time.

Johnson's three final conclusions seem more than the evidence justi-

fies. We know only that the deep sands of the Appalachian and a few other fields generally are dry. We do not know that the shales are dry. In fact, I suspect that the shales are moist, since the sands still contain water in places. Hence it does not follow that the very deep oil of such fields is scattered in the shale, largely beyond recovery. The value of the very deep reserves is diminished, *first,* by the expense of deep drilling, and *secondly,* by the lack of sufficient water in the sands to control the accumulation of the oil in definite structures. The few definitely synclinal fields, all in dry sands, as near Bluff, Utah, also in southwestern Wyoming and southern Kentucky, have not been very successful. To produce a profitable accumulation of oil *in a continuous sand,* the sand must contain enough water to greatly restrict the oil-bearing area.

Deep drilling for oil admittedly is a failure in the Appalachian (or Eastern) fields of the United States; but in California, where the deep sands contain much more water, there is good hope of developing very deep fields. Even now there are a few California wells producing oil from below 4,000 ft.

The two remaining conclusions of the author rest on similar evidence and are subject to the same criticism. The general existence of large deep reserves of natural gas cannot be predicted from the available evidence. Possibly the deep gas may be badly contaminated with abyssal nitrogen, as appears to be the case in the western fields of Kansas. The peridotite dikes of western and of central Kentucky, and the more widely distributed barite-fluorite deposits of the same regions, indicate that abyssal emanations may be important in the deep strata of the sub-Appalachian oil and gas belts. These are interesting possibilities. The evidence to be obtained in deep wells offers great promise of contributions to geological knowledge, in fundamental problems that have received little attention. However, there is little justification for general economic predictions concerning the zone immediately beneath that now explored by the drill.

The Petroleum Fields of Alaska*

BY ALFRED H. BROOKS, WASHINGTON, D. C.

(New York Meeting, February, 1915)

Introduction

PETROLEUM seepages are known in Alaska at four localities, all on Pacific seaboard. These, named from east to west, are Yakataga, Katalla on Controller Bay, Iniskin Bay on Cook Inlet, and Cold Bay on the Alaska Peninsula. Besides these, a petroleum residue has been found near Smith Bay on the north Arctic coast of the Territory. At Katalla, Cold Bay, and Iniskin Bay there has been some drilling for oil, and in the first field several productive wells have been opened up. Alaskan petroleum, so far as its composition is known, is a refining oil, with paraffin base and low sulphur content.

Most of what is known of the geology of Alaska oil fields is based on the investigations of Dr. George C. Martin, of the United States Geological Survey. A. G. Maddren, of the same service, has recently made a reconnaissance of the Yakataga oil field. The data to be here presented are chiefly taken from the following publications:

G. C. Martin: The Petroleum Fields of the Pacific Coast of Alaska, with an account of the Bering River coal deposits, *Bulletin No.* 250, *U. S. Geological Survey*, pp. 9 to 27 (1905).

G. C. Martin: Geology and Mineral Resources of the Controller Bay Region, Alaska, *Bulletin No.* 335, *U. S. Geological Survey*, pp. 112 to 130 (1908).

G. C. Martin and F. J. Katz: A Geologic Reconnaissance of the Iliamna Region, Alaska, *Bulletin No.* 485, *U. S. Geological Survey*, pp. 126 to 130 (1912).

A. G. Maddren: Mineral Deposits of the Yakataga District, *Bulletin No.* 592, *U. S. Geological Survey*, pp. 143 to 147 (1914).

Katalla Field

The Katalla field is marked by a series of seepages and gas springs occupying an east and west belt, about 25 miles in length and from 4 to 8 miles wide. (See Fig. 1.) This zone skirts the north shore of Con-

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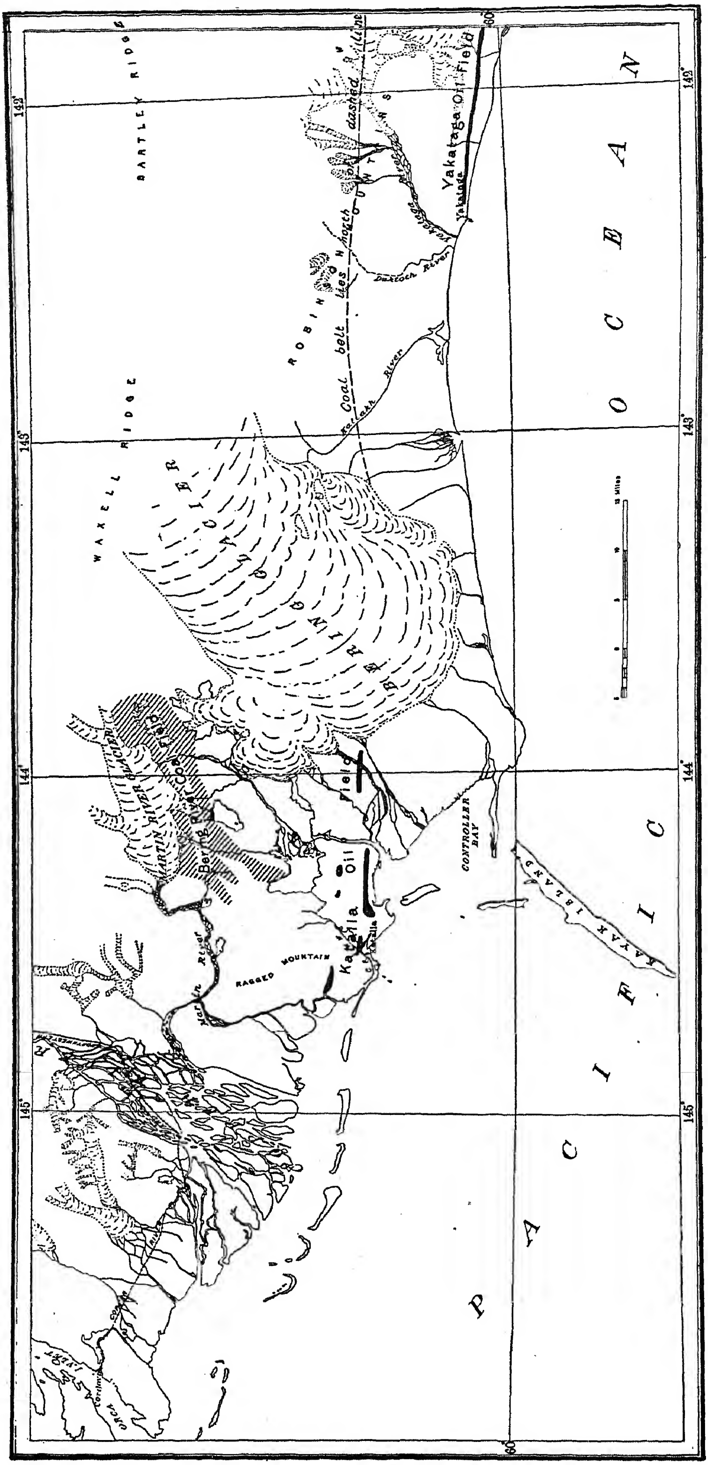

FIG. 1.—MAP SHOWING LOCATION OF KATALLA AND YAKATAGA OIL FIELDS, ALASKA.

troller Bay. To the east it extends into the alluvial flats of Bering River and to the west into the flats of Copper River. It lies, in part, in the southern slope of a densely timbered highland whose summits reach to 1,200 or 2,000 ft. above the sea; in part, in the flats adjacent to the shore line. Drilling has been done at several localities in this belt, but the productive wells are limited to that part of the belt lying between the town of Katalla and Bering River.

The surface rocks consist chiefly of a series of intensely folded and faulted shales, sandstones, and conglomerates, with some small basalt or diabase dikes and sills. The general structural trends are about N.20°E., and the line of seepages lies diagonal to this structure. As much of the field is masked with a dense growth of vegetation, details are lacking on the minor structures. In certain instances, seepages or groups of seepages are closely associated with faults or with subordinate anticlines. It is true, however, that no definite law of occurrence of the petroleum with tectonic features has yet been established. West of Katalla seepages have been found belonging to the same belt, in surface rocks of more or less metamorphosed graywackes and slates.

The shale, sandstone, and conglomerate series is of Tertiary age and probably older than the Tertiary coal measures of the Bering River field which lies about 25 miles to the northeast. All these Tertiary beds are undoubtedly younger than the metamorphosed graywackes and slates, which may be of Mesozoic or of Paleozoic age.

It will be evident from the above discussion that the source of the oil is unknown. A source in the metamorphic graywackes is, of course, very improbable. It may be derived from the Tertiary beds or from strata not exposed at the surface. A possible explanation of the seepages in the metamorphic rocks is that these have been thrust over younger oil-bearing horizons.

Katalla, the distributing point for this field, is a small settlement at which freight can be landed from scows only during favorable conditions of the wind. During the oil excitement some use was made of Controller Bay, 15 miles east of Katalla. Within its shelter ships discharged on scows, and these were landed at the mouth of Bering River. Plans have been formulated for developing the Bering coal field by a branch from the Copper River Railroad, connecting with tidewater at Cordova on Prince William Sound. (See Fig. 1.) Another plan contemplates a railway stretching inland from a terminal on Controller Bay. Either plan could be made to serve the Katalla field with but little additional expense. There is ample timber available for structural purposes.

Yakataga Field

The Yakataga petroleum field lies about 80 miles east of Katalla. Here a series of seepages marks a zone about 20 miles in length and one-half

to two miles from the beach. The extension of this line to the west carries it into the Pacific, and to the east into an unexplored and ice-covered region tributary to Icy Bay. Prospectors report the presence of a strong seepage near Yahtse River, about 15 miles east of the locality to which the belt has been actually traced. There is also a less definite report of the occurrence of seepages along the mountain front between Yakutat and Lituya Bay, about 200 miles east of Yakataga. What little is known of the geology of this region lends some support to this rumor.

The line of Yakataga seepages lies for the most part in a series of short valleys separated from the coastal plain by a low wooded ridge and drained by streams whose courses are transverse to this ridge. About a dozen seepages have been found, most of which are little more than exudations along joint cracks. One, however, on Johnston Creek is roughly estimated to discharge a barrel or more of petroleum a day.

So far as determined, all these seepages lie along a sharp anticline whose southern limb is about vertical and whose northern limb dips inland at from 15° to 45°. The exposed rocks consist of sandstone overlain by fine-textured shale of Oligocene or lower Miocene age. No drilling has been done in this field, so that there is no information at hand of the sub-surface geologic conditions. Speculation as to the source of the petroleum is, therefore, futile.

The Yakataga district is to-day almost inaccessible, as all landings must be made on a beach exposed to the full sweep of the Pacific. Most of the freight for the few placer miners in the district is brought from Katalla by launch when the winds are favorable to a landing. The overland route along the beach is difficult, though it probably could be utilized for a pipe line or narrow-gauge railroad. The marked recession of a glacier at the eastern end of the field during the past few years has revealed a small indentation known as Icy Bay in which shelter may possibly be had, though at present it is in part blocked by ice cakes. If the glacier continues to retreat, the Yakataga field will probably be rendered accessible. There is abundant timber suitable for structural purposes in the district.

Iniskin Bay Field

Iniskin Bay is an indentation about 12 miles deep which, with Chinitna Bay on the north, blocks out an irregular-shaped peninsula on the west shore of Cook Inlet. The more or less even shore line of this peninsula is broken on the southwest by two small indentations—Oil Bay and Dry Bay. Petroleum seepages have been found in this field near Iniskin, Oil, and Dry bays.

The bedrock of the field is a fine-grained sandstone with which are interbedded some clay shales. Some beds of conglomerate occur in the

sandstone, one of which forms the basal member of the formation, and near the head of Iniskin Bay this rests on sheared igneous rocks. The sandstone and the associated sediments, which are of Middle Jurassic age, have a thickness of about 1,100 ft. It is overlain by a shale formation with intercalated conglomerate. The seepages occur in the eastern limb of a broad anticlinal arch which has been somewhat faulted. Drilling has not gone deep enough to indicate whether the altered igneous rocks underlie the whole field as they do at the western margin. The evidence in hand points to the conclusion that the sandstone is the source of the petroleum.

The field is readily accessible from the good harbors lying both north and south. These are occasionally blocked by ice floes, but are usually accessible throughout the year. There is some timber in the district, but this is chiefly of an inferior quality. Timber could, however, be brought from other parts of Cook Inlet region at no great expense.

Cold Bay

Cold Bay is an indentation on the Pacific shore of the Alaska Peninsula nearly opposite the south end of Kodiak Island (Fig. 2). The adjacent area, which is untimbered, consists of rounded hills rising to altitudes of less than 1,000 ft. Above this altitude rise some higher peaks, chiefly made up of volcanic rocks.

The country rock of the area in which the seepages occur is a sandstone and shale formation carrying a little limestone; is of Middle Jurassic age, and about 2,000 ft. in thickness. It is underlain by shales, limestones, and cherts (Triassic) and succeeded by Middle Jurassic arkose conglomerate, sandstone, and shale. The youngest rocks of the district are volcanics, chiefly andesites and basalts. It is a significant fact that the oil-bearing member of this series is of the same age as the oil-bearing horizons of the Iniskin Bay field. The main structural features are broad open folds whose axes parallel the coast, trending about northeast and southwest. The dips of the strata seldom exceed 15°. Some faults have also been noted in the district. There are a number of oil seepages in the field, some of which are strong. At one of these seepages there is considerable flow of gas.

There is a good harbor at Cold Bay open throughout the year, and the contour of the region makes it readily accessible. There is no timber in the district.

Smith Bay Locality

E. de K. Leffingwell, who for a number of years has been exploring the north coast of Alaska, has reported the occurrence of petroleum residue about 100 miles east of Point Barrow. Mr. Leffingwell describes this

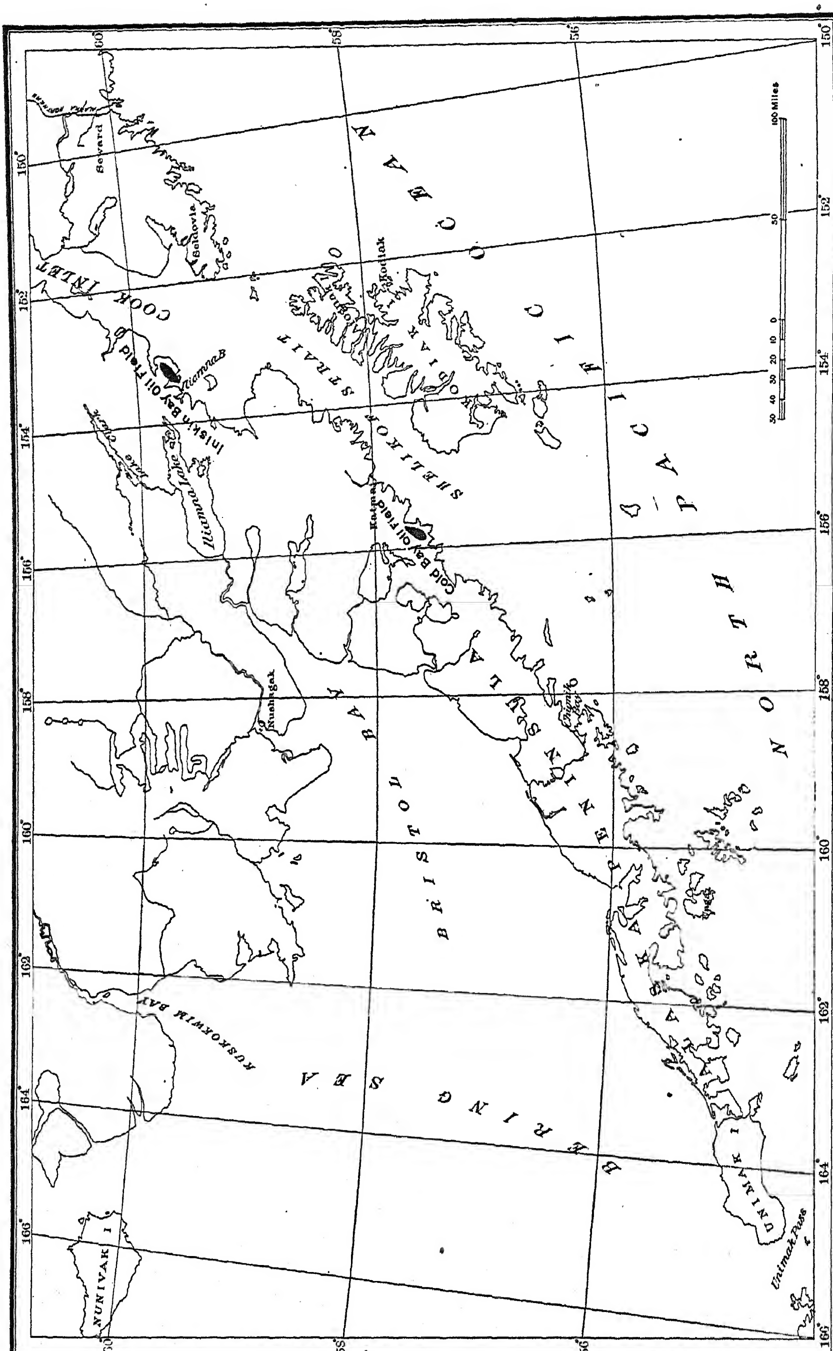

FIG. 2.—MAP SHOWING LOCATION OF INISKIN BAY AND COLD BAY OIL FIELDS, ALASKA.

material as occurring near Smith Bay[1] in a mound several hundred yards in diameter and standing about 150 ft. above the level of the tundra. It contains considerable vegetable matter and silt, but appears to be the residue of a petroleum containing an asphaltic base. Nothing is known of the geology at the locality of the find, but in the general province in which it occurs there is a series of horizontal Tertiary sediments overlying gently folded Mesozoic sediments. Even if an oil pool were found in this northern field the conditions of transportation would prohibit present commercial development. Smith Bay is a shallow indentation and is locked in the ice at least 10 months in the year.

Character of the Petroleum

The Alaskan petroleum, so far as its quality is known, is a refining oil similar to that of Pennsylvania, and has, as already stated, a high percentage of volatile compounds, a paraffin base, and contains but little sulphur. The accompanying table will serve to illustrate the quality of the oils from the Katalla and Yakataga fields, and, so far as known, those from the Iniskin and Cold Bay fields have about the same composition. It should be noted that all the samples from the Yakataga field and part of those from the Katalla field are from seepages where there has been a loss of the volatile compounds.

Development

The presence of petroleum seepages on the Alaska Peninsula seems to have been known to the Russians as early as 1865, but aroused no interest. Oil was reported along the Pacific seaboard in 1880, but it was not until about 1896 and 1897 that definite attention was drawn to the Katalla and Yakataga fields. The first drilling was done at Katalla in 1901. Meanwhile a survey was made for a pipe line from Controller Bay to the Yakataga field, but no drilling has been done in the latter field. In 1901, the first drilling was done in the Iniskin field, and two years later three wells were sunk in Cold Bay.

The enormously rapid increase of the oil output from the California fields caused a sudden collapse of the Alaska oil boom, which reached its zenith in about 1904. Operations in Alaska were very expensive, largely on account of cost of transportation, and, in spite of the high grade of the petroleum found, operators were soon discouraged. Moreover, at the time of the petroleum excitement the local market for the product was inconsiderable. In 1906, all Alaskan petroleum lands were withdrawn from entry, which served as a further deterrent to operations.

[1] Brooks, Alfred H.: The Mining Industry (Alaska) in 1908, *Bulletin No.* 379, *U. S. Geological Survey*, pp. 61, 62 (1909).

Summary of Analyses and Tests of Katalla and Yakataga Petroleum

Locality	Color	Specific Gravity	Gravity Degrees Baumé	Flashing Point, Degrees F.	Benzine Per Cent.	Kerosene, Per Cent.	Lubricating Oil, Per Cent.	Residue Coke and Loss, Per Cent.
KATALLA FIELD								
Katalla, well, 10[a]...	...	0.8280	39.1	...	21.0	51.0	28.0	...
Katalla, well, 10[b]...	...	0.7958	45 9	...	38.5	31.0	21.5	9.0
Katalla, well, 10[c]...	Light green.	0.7957	45.9	70–80	38.5	31.0	21.5	9 0
Katalla, well, 10[c]...	...	0 800	...	...	34.2	34.4	16 5	14 5
Katalla, well, 10[d]...	Dark red...	0.802	...	−60	...	...	...	...
Katalla, well, 10[d]. .	Dark red...	0.790	...	−60	...	...	...	...
(?)[d]...	...	0.869	...	...	...	19.0	78 6	1.7
(?)[d]...	...	0.914	...	...	...	9 0	87.6	2.7
(?)[d]...	...	0.800	...	...	24 8	53.9	16 7	1.2
Katalla Meadow [d]..	Dark brown	0.929	...	240	...	...	...	...
Katalla Meadow [d]..	...	0.901	...	156	...	...	...	...
Katalla Meadow [d]..	...	0.874	...	156	...	...	...	...
Katalla Meadow [d] .	...	0 869	...	152	...	...	...	...
Katalla Meadow [d]..	...	0.961	...	266	...	...	...	...
Burls Creek [d]...	Dark red-brown.	0.942	...	234	...	...	...	...
YAKATAGA FIELD								
Johnston Creek, [d, e].	Dark brown	0 964	...	200	...	...	...	...
Johnston Creek, [d, e].	Dark brown	0.879	...	178	...	...	...	...
Poule Creek [d, e]...	Dark brown	0.970	...	250	...	...	...	...
Poule Creek [d, e]...	Dark brown	0.881	...	67	...	...	...	...
Poule Creek [d, e]...	Dark brown	0.914	...	156	...	...	...	...
Crooked Creek [d, e]...	Dark brown	0.921	...	172	...	...	...	...
Oil Creek [d, e]...	Dark brown	0.855	...	108	...	...	...	...
Yakogelty [d, e]...	Dark brown	0.937	...	246	...	...	...	...
Morrison Creek [d, e].	Dark brown	0.991	...	270	...	...	...	...
Argyll Creek, Icy Bay [d, e].	Dark brown	0.962	...	310	...	...	...	...

Bulletin No. 592, *U. S. Geological Survey*, p. 146 (1914).

[a] Sample collected by G. C. Martin, test by Penniman and Browne. *Bulletin No* 335, *U. S. Geological Survey*, p. 121 (1908).

[b] Oliphant, F. H.: The Production of Petroleum in 1902, *Mineral Resources*, 1902, *U. S. Geological Survey*, p. 583 (1903).

[c] Stoess, P. C.: The Kayak Coal and Oil Fields of Alaska, *Mining and Scientific Press*, vol. lxxxvii, p. 65 (1903).

[d] Redwood, Boverton, Petroleum, vol. i, 2d ed., p. 198 (1906).

[e] The exact localities of seepages where these samples were taken are not known, but they are believed to be in the Yakataga field.

Assessment work has, however, been kept up on some claims staked previous to the withdrawal. Patent has been granted to a small group of claims in the Katalla field, which constitute the only Alaskan oil lands to which the government has relinquished title.

In all, about 26 holes have been drilled in the Katalla field, of which probably 10 have struck some oil. The deepest well is about 1,600 ft., but the geology is so complex that the actual depth is not significant of the position of an oil pool. Some natural gas was encountered in the drilling. An output of about 2 to 10 bbl. is reported from the productive

wells. In 1912, a small refinery was erected near Katalla. Since then this plant has supplied gasoline, illuminating oil, etc., to various local industries.

In the Iniskin Bay field one well was sunk to about 1,000 ft., encountering gas and oil. A strong flow of water shut off the oil. A second well was abandoned at a depth of 450 ft. A third well met with oil and gas at a depth of 770 ft., and the hole was continued to a depth of 900 ft. There has been no drilling in this field since 1904.

In the Cold Bay field two wells have been drilled to depths of about 1,500 ft., and several oil sands are reported. There have been no developments since 1904.

Summary and Conclusions

The Katalla field is a region of closely folded and faulted Tertiary rocks, possibly comparable in its structure to that of the California fields. Much of the drilling in this field was done without any regard to geologic structures, and but few good logs are available. The presence of oil is indicated by the success achieved, but it will require further drilling to determine the presence or absence of a large pool. At Yakataga the structure of the oil-bearing formation appears to be much simpler, and it ought to be possible to test the field without a very large amount of drilling.

In both the Iniskin and Cold Bay fields Middle Jurassic sandstone forms the country rock in the oil-bearing area. The open folding should be favorable to oil pools; on the other hand, some faulting has been observed. There is reason to believe that the oil-bearing formation found in these two fields may have a wider distribution in the Alaska Peninsula. Whether it carries petroleum at other localities has not been determined. On the whole, there is good hope of finding petroleum in the Alaska Peninsula.

Alaska affords a steadily growing market for petroleum and petroleum products. In 1913, 15,682,000 gal. of crude oil, 1,735,000 gal. of naphtha, 661,000 gal. of illuminating, and 150,000 gal. of lubricating oil were shipped to Alaska. If any oil field were developed on the Alaska Peninsula, it would be specially well located for supplying the local markets on Seward Peninsula, Yukon River, and Pacific seaboard.

A Modern Rotary Drill

BY HOWARD R. HUGHES, HOUSTON, TEXAS

(New York Meeting, February, 1915)

In drilling for water and oil to reasonable depths through the generally soft yielding clay and sand formation of the Coastal Plain of Texas, Louisiana, and Mississippi, the rotating method of drilling was adopted, principally on account of the easy and quick penetration, and the low cost of the drilling plant.

In favorable ground, free of heavy gravel and rock strata, as much as 1,000 ft. has been drilled in less than 36 hr., although such performances were of course rare.

Hydraulic rotary drilling, or, as it is now called, rotary drilling, was used in the above States as early as 1880. The plant consisted of an ordinary derrick, a 25-h.p. boiler, a small hoist, a steam pump, and a water swivel with hose attachment, and an ordinary flat diamond-pointed bit.

The successful drilling in of a phenomenal oil well by this process on Spindletop, near Beaumont, Texas, on Jan. 10, 1901, and the ascertained impracticability of drilling subsequent wells in the same locality by other methods (owing principally to heavy quicksand under pressure from below), brought rotary drilling into great prominence, practically to the exclusion of any other process throughout the Coastal Plain, and later on elsewhere.

The method, as its name implies, involves the rotation of a pipe by means of machinery placed on the derrick floor. A drill bit attached to the lower end cuts a clearance for the drill pipe, with much the same motion and effect as an auger. Water forced through the drill pipe by means of a pump, and escaping through the bit, removes the cuttings and returns to the surface outside the drill pipe. In this manner the hole is kept open, permitting the drill stem to rotate freely.

The pressure of the column of muddy water holds up the walls of the hole until it has been cased.

Practically all the wells of the Gulf coast region, numbering nearly 10,000, have been drilled with this system. During the last five years its use has been extended to many other States and countries.

The great drawback hitherto has been the slow progress made in drilling rock and other hard formations. For soft, caving formations no other system can approach it in efficiency.

The old style bit in general use is known as the fishtail type, as shown in Fig. 1. Having only two cutting edges it soon grinds down flat when hard rock is encountered. At times only a few inches a day can be made with it. The racking and wrenching to which the machinery and drill

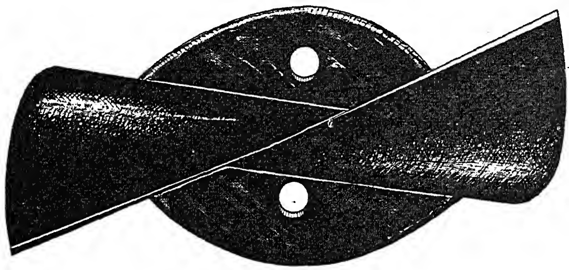

Fig. 1.—Fishtail Bit.

pipe are subjected, when drilling a hole from 4 to 18 in. in diameter, result often in twist-offs of the drill stem and costly fishing jobs. With the use of the heavier rigs and deeper drilling the need for a bit adapted to cutting rock became a crying necessity.

It was to meet this need that the cone bit, known as the Sharp & Hughes, was invented by me in 1908.

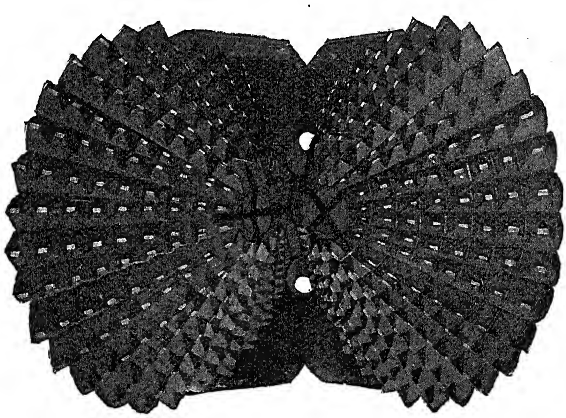

Fig. 2.—Sharp & Hughes Cone Bit.

In brief, it consists of two or more detachable, cone-shaped cutters of hardened steel (see Fig. 2). These cutting cones revolve on bronzed bearings, lubricated with a special heavy viscous oil supplied by means of a small pipe carried inside the drill stem (see Fig. 3). The cutters, being detachable, may be removed and sharpened when dull.

The old style fishtail bit scraped its way through the rock encountered. With hard rock or sandstone it soon wore flat, lost all cutting power and had to be renewed. This necessitated the removal and replacement of the entire drill stem, with many (perhaps several thousand) feet of pipe, the work of many hours.

The principle of the cone bit is entirely different. The edges, or cone points of the bit, roll in a true circle like a cone bearing, and crumble or chip away the rock. The cone points, being of very hard steel, wear away slowly. Often they show but slight wear after drilling 50 ft. of rock, a few inches of which would completely dull the ordinary fishtail

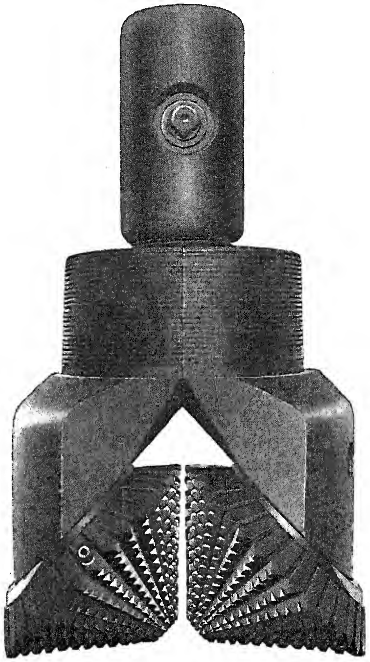

Fig. 3.—Broadside View of Sharp & Hughes Cone Bit.

bit. The rolling motion allows the cutting edges on the cones to chip the rock, one edge after another.

Fig. 4 shows the bit, drill pipe, and lubricator in the hole, ready for drilling. The lubricator pipe, about 12 ft. long, is filled with oil, which is forced down into the bit by the pressure of the circulating water on the plunger. This figure shows also that the bottom of the drill hole as formed by the operation of the bit presents a perfect seat for a water-tight joint, preventing leakage after the casing has been set. When the cone bit is introduced in a hole to which previous use of the fishtail or diamond-pointed bit has given a V-shaped bottom, it must be advanced slowly

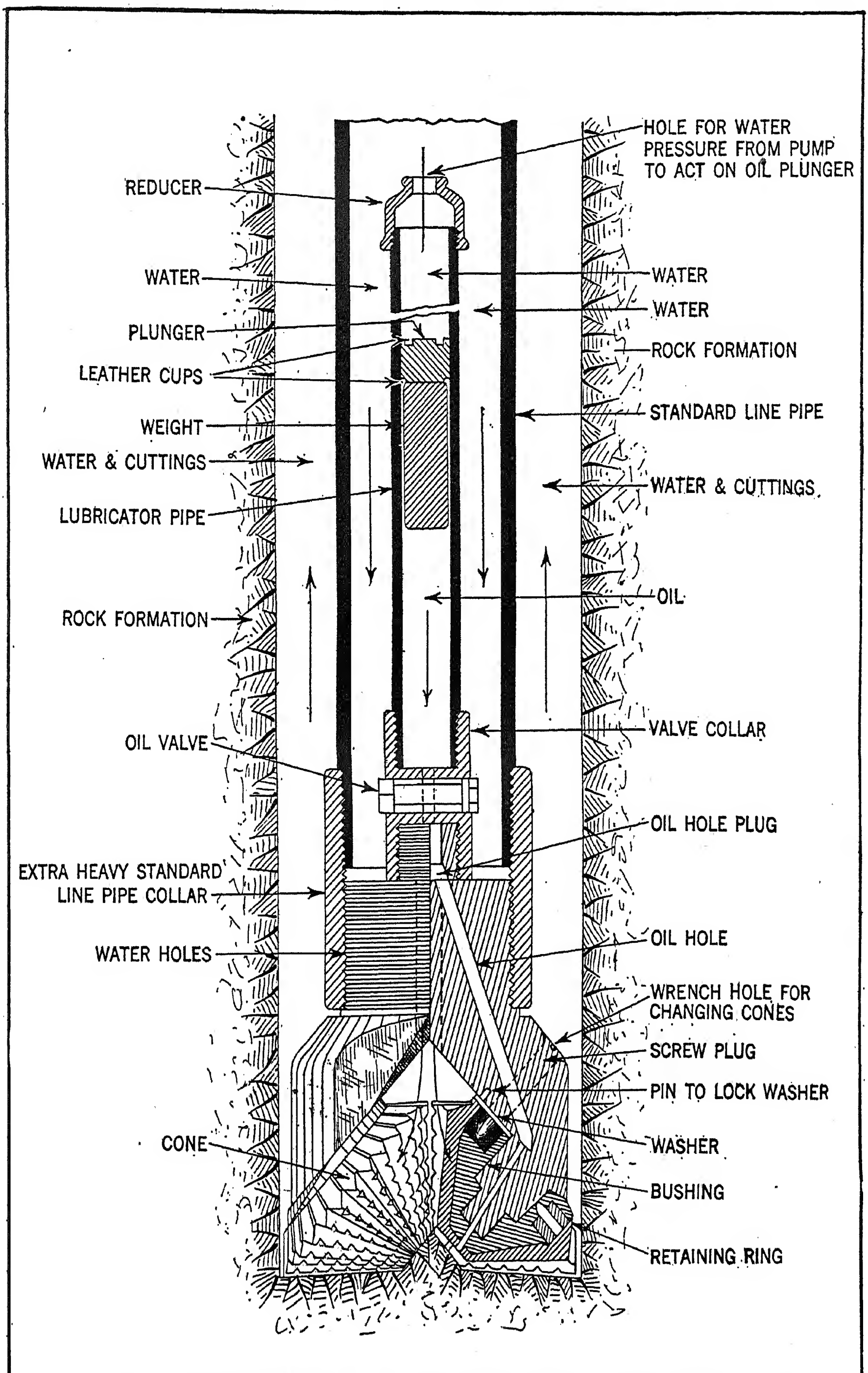

Fig. 4.—Sharp & Hughes Cone Bit Ready for Operation.

and carefully for the first foot, so that it may change that shape to suit its own form of cut.

Fig. 5 shows the whole drilling-plant and apparatus of this system.

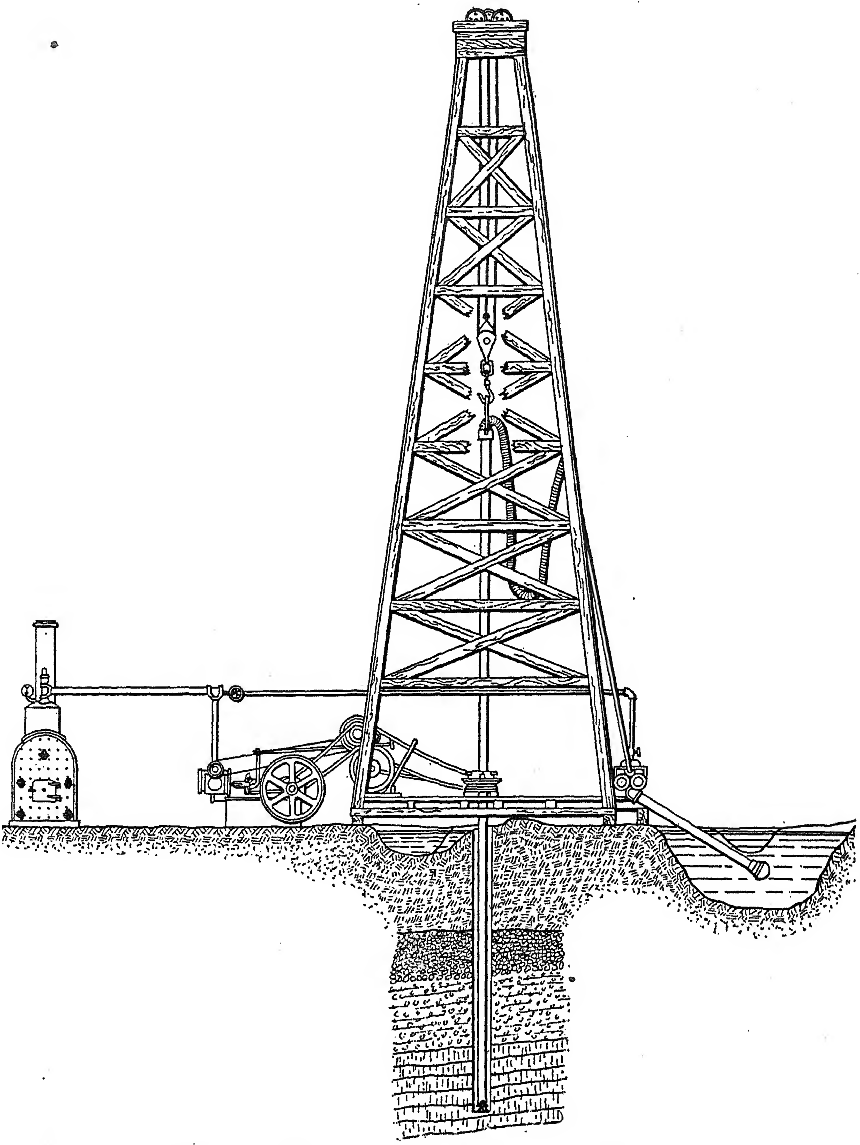

FIG. 5.—A COMPLETE ROTARY DRILLING OUTFIT.

The proper adjustment of the weight upon the bit is the secret of good work with this drill. Experience has shown that the following weights give satisfactory results for the corresponding bits of standard sizes:

Drilling Weight on Bit

Diameter of Bit, Inches	Weight, Pounds	Diameter of Bit, Inches	Weight, Pounds
2⅜	2,870	6½	7,900
2⅞	3,480	7⅛	8,650
3⅞	4,700	7⅝	9,270
4⅜	5,310	7⅞	9,560
4⅞	5,910	8½	10,300
5½	6,670	9	11,000
5⅞	7,140	9⅞	12,000

For bits larger than those in the table, as much weight as practicable may be employed with little or no risk of overloading the bearings in the bit. But within the limits of the table, the weights given are probably as great as prudence would permit. For extremely hard rock, the speed, not the weight, should be increased.

The following actual working tests show the performance of the cone bit in comparison with the old fishtail:

The House No. 1 well of the Producers' Oil Co., Humble, Texas, struck rock at 1,819 ft., after which the fishtail bit bored 38 ft. in 19 days—an average of 2 ft. a day. The cone bit was then substituted, and bored 72 ft. in 6 days, or 12 ft. a day.

The comparative costs were as follows:

FISHTAIL

Day crew:	
4 Men, $3 per day, 19 days	$228.00
1 Driller, $200 per month, 19 days	126.67
Night crew:	
4 Men, $3 per day, 19 days	228.00
1 Night driller, $5 per day, 19 days	95.00
Dressing 38 9-in. fishtail bits at $2.25	85.50
Steam and water, 19 days, at $10	190.00
Total cost for 38 ft. of 9-in. hole	$953.17
Average cost per foot	$25.08

HUGHES CONE BIT

Day crew:	
4 Men, $3 per day, 6 days	$72.00
1 Driller, $200 per month, 6 days	40.00
Night crew:	
4 Men, $3 per day, 6 days	72.00
1 Night driller, $5 per day, 6 days	30.00
Steam and water, 6 days, at $10	60.00
Rental of bit for 30 days	50.00
3 Sets of cones, at $45	135.00
Total cost of 72 ft. of 9-in. hole	$459.00
Average cost per foot	$6.38

This comparison takes no account of the strain and wear upon machinery, the greater cost of superintendence per foot drilled, and the loss of time, under the old system.

While primarily designed for oil and water wells, this cone bit can be applied in drilling sump holes for mine pumps, in making air shafts, and in driving holes inclined at various angles from the vertical. Its use greatly enlarges the field of the rotary system, and the cone bit already is extensively used in California, Mexico, Trinidad, Roumania, Russia, Persia, Egypt, Japan, Borneo, and India, by presenting the great advantages of more rapid drilling through hard rock; of reaching greater depths than any other rotary apparatus can compass; of finishing a hole with smaller reduction of surface diameter than any other system permits; of the consequent requirement of fewer "strings" of casing; of less deterioration of drill pipe through strains and vibration, and of the saving of much time consumed in removing the drill pipe for sharpening or changing bits.

Discussion

I. N. Knapp, Ardmore, Pa.—The claims made in the paper for the Sharp & Hughes cone bit are modest. I have used it with satisfactory results. It is a wonder for eating through hard, brittle rock; it will drill through iron pyrites that the fishtail bit cannot enter.

Soft ground is apt to contain concretions that are much harder than the containing ground. We have at times in drilling with a fishtail bit thought the hole was partly in such hard material and partly in soft, thus making a difficult place to drill past in the ordinary way. By changing to a Sharp & Hughes cone bit such obstructions were easily overcome. This cone bit has greatly widened the range of work that may now be profitably accomplished by the hydraulic rotary method of drilling.

It is not, however, adapted to drill soft or sticky ground, the old fish-tail bit being better for such purposes. The fishtail bit usually scrapes its way through the material drilled, but not always. I believe at times it cuts and chips in soft friable rock. That is, the bit seems to bite into the rock, and when sufficient torsion is put on the drill stem it breaks its hold and jumps around and chips the rock from the bottom of the hole, like a stone cutter's chisel, and then takes another hold. This condition occurs only in friable material when the bit is sharp and the feed properly controlled. The Sharp & Hughes cone bit is a most valuable tool for the rotary driller.

The Dehydrating Oil Plant of Nevada Petroleum Co., California

BY S. J. HARDISON, COALINGA, CAL.

(New York Meeting, February, 1915)

In the fall of 1912, the appearance of water in the oil of the Nevada Petroleum Co., Coalinga, Cal., made necessary the installation of a dehydrating plant to reduce the water below the 3 per cent. limit prescribed by the agency.

Unlike the mining industry, technical literature of the oil industry is limited and extremely unsatisfactory. Until the recent efforts of the Petroleum and Gas Committee of the American Institute of Mining Engineers, no concerted movement has been instituted to secure publication of papers dealing with the problems of the oil business, and because of the additional fact that dehydrating of oil has not been practiced in California to any extent up to comparatively recent time, it was found necessary to experiment in order to determine the most satisfactory plan for this purpose.

It is easy to write regarding successful enterprises, but, while not so pleasant, it is equally desirable to write of failures so that others may be saved the loss incident to such investments.

The water in this oil occurs both free and as an emulsion. Free water easily settles out, but the emulsion requires treatment. So far as can be determined, the emulsion consists of globules of water surrounded and enveloped by a film of oil. Starting with this hypothesis the theory has been evolved that in order to break up this emulsion the water must be heated at least to boiling point, when an explosion takes place destroying the globule. Unfortunately, this condition was not recognized in the installation of the first plant, which was planned on the following lines:

A heater was arranged so that all the oil from the wells could be heated before reaching the shipping tanks, these being already fitted with coils for steam heating. Tanks were filled with about 4 ft. of water, live steam was turned into the coils and the water brought nearly to the boiling point, when the heated oil was run in. This for a time was partly successful, but the length of time necessary to apply this heat was too great for practical operating.

The arrangement of the heater plant was as shown in Figs. 1, 2, and 3. Three old boilers were suspended on substantial pipe supports, dry

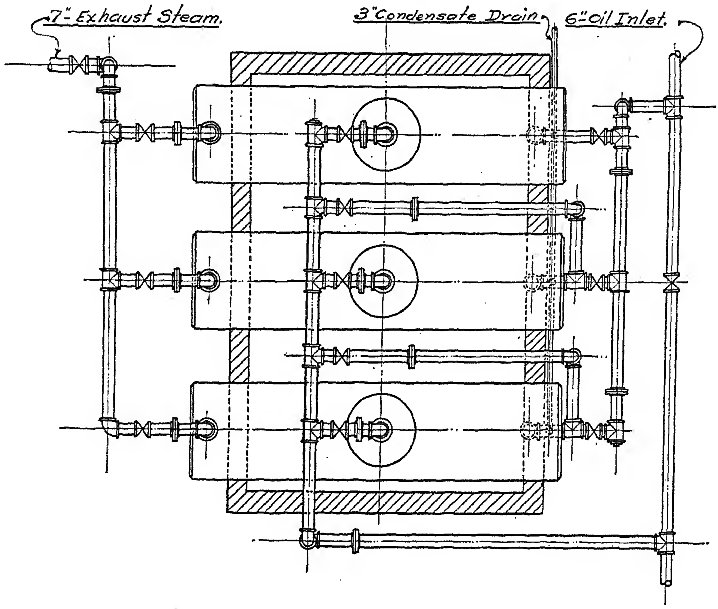

FIG. 1.—FIRST OIL-HEATING PLANT. PLAN.

brick walls filled up with oil sands being used to prevent radiation on bottom and side of boilers; the tops of the boilers were covered with bats and oil sand after the pipe work was done. Headers of galvanized

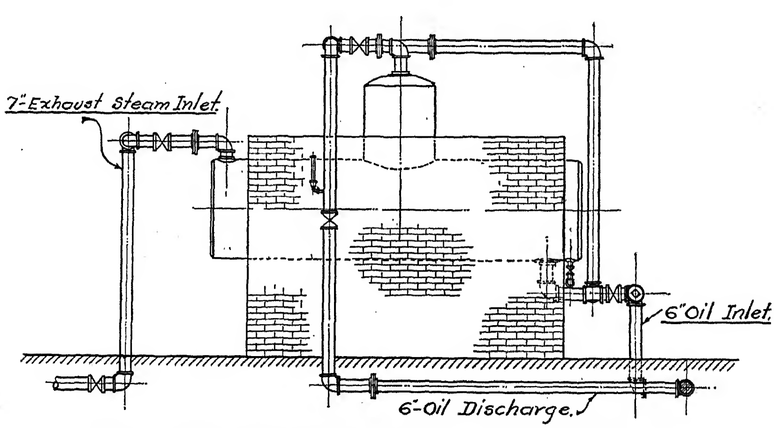

FIG. 2.—FIRST OIL-HEATING PLANT. SIDE ELEVATION.

iron were made to connect the inlet and the outlet steam. Exhaust steam from the compressor and pumps of the central plant was carried

through 7⅝-in. casing pipe, connected to the header and thus through the boiler tubes. The outlet was high enough for the condensed steam to gravitate back to the hot-water well of the central plant.

The plant was placed near the shipping tanks and close to the main line carrying oil from the wells to the shipping tanks.

The piping was arranged so that the oil could be run from the main through one, two, or all three boilers, usually traveling through all three; or the oil could be run direct through the line as before, in case of leakage in the plant. As no fire was used around this plant, there was no danger from that source.

This plant heated the oil to an average temperature of 170° F. very satisfactorily, and with practically no attention or cost, outside of installation. The loss of about 10° in transit to the tanks made about 60° to be supplied by the live-steam coils in the tanks. The water-heating

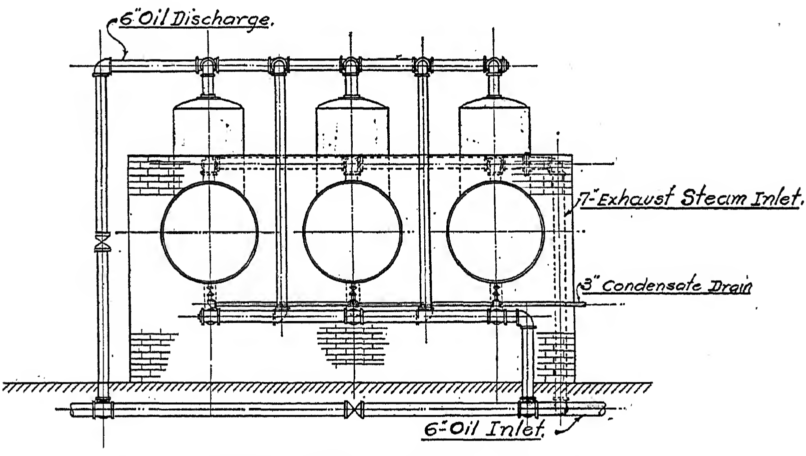

Fig. 3.—First Oil-Heating Plant. End Elevation.

system was not adapted to the conditions and was discarded, the heat then being applied directly to the oil through the coils, but this was also too slow for the necessary operating capacity.

It is certain that the failure of this plant was due to insufficient heating of the oil, and a second plant was built with a view to correcting this difficulty. This plant was constructed as follows:

Two old 40-h.p. boilers were erected, with the typical oil-field mud setting, the boilers being suspended on pipe supports, with regular oil burners for supplying heat direct under boilers.

The piping was arranged so that oil could be run through either or both boilers, the oil being run in at the bottom and rear, and out at the dome; and again in at the bottom of the second boiler, through it and out to the storage reservoir, from whence it was pumped to the shipping

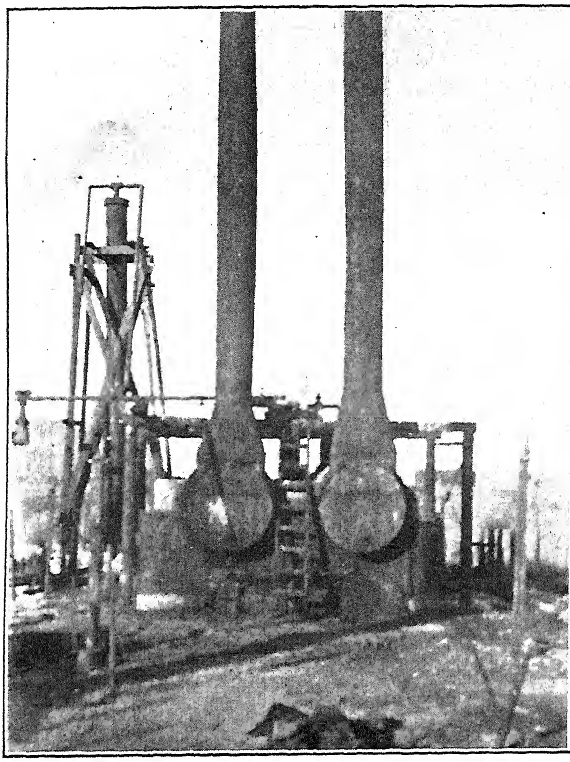

FIG. 4.—SECOND OIL-HEATING PLANT.

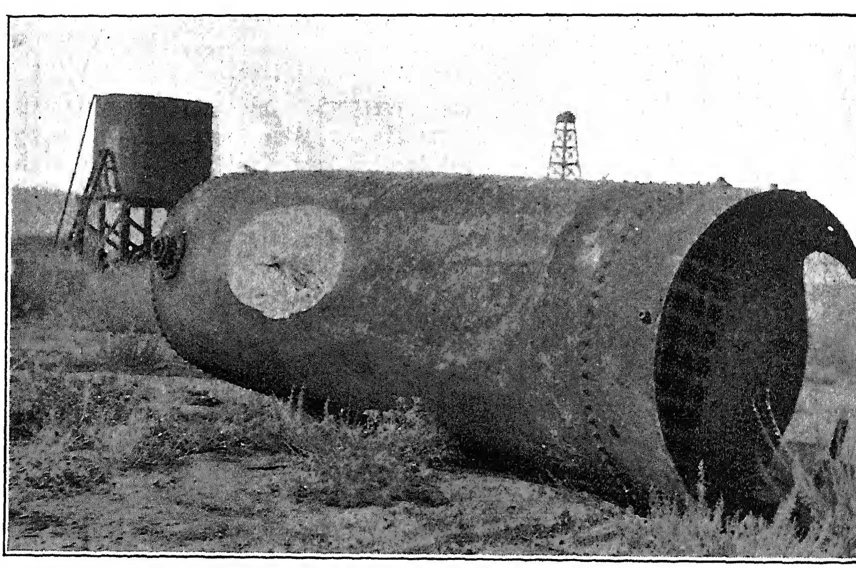

FIG. 5.—BURNT-OUT BOILER, SHOWING BAGGING.

tanks. The heater was still used for this plant, and when both boilers were being used a light fire was carried under the first boiler, the heavier fire being applied to the second boiler. The 6-in. overflow delivery

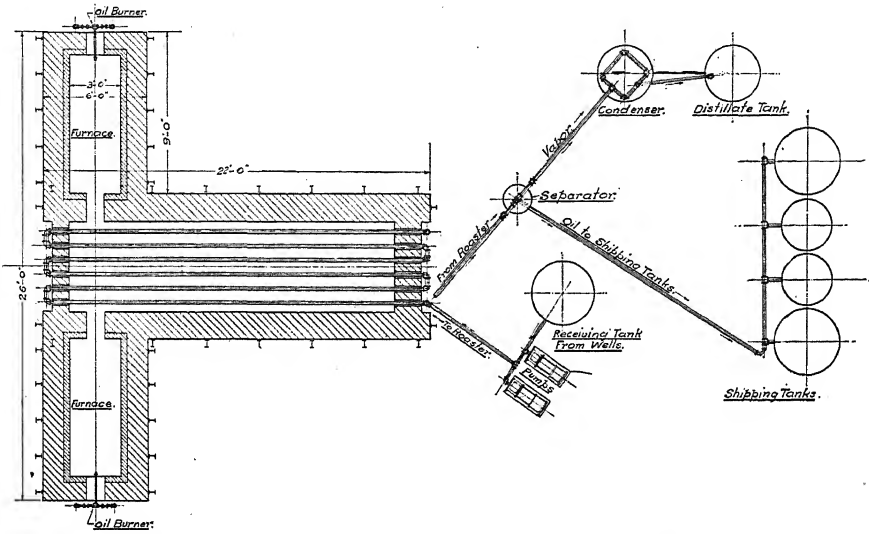

FIG. 6.—OIL-DEHYDRATING PLANT OF NEVADA PETROLEUM CO., COALINGA, CAL. PLAN.

pipe was set on a slight up grade to the receiving tank at the reservoir to insure a back pressure on the boiler.

The gauges usually showed a pressure of a few pounds and a tem-

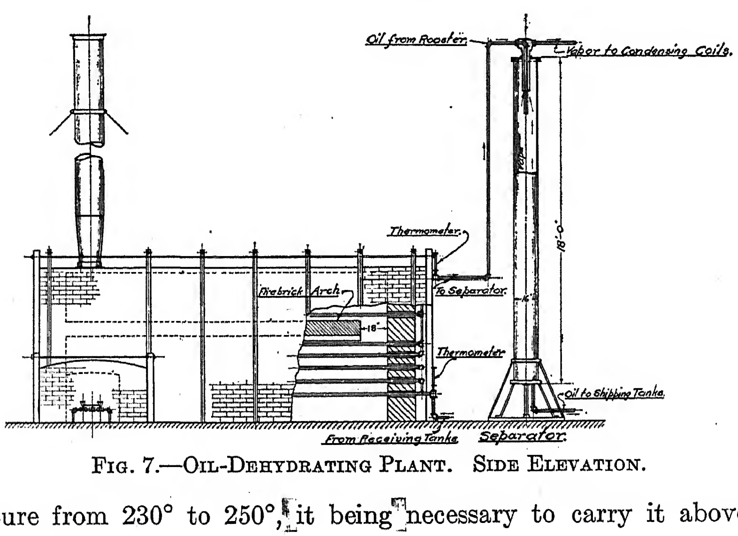

FIG. 7.—OIL-DEHYDRATING PLANT. SIDE ELEVATION.

perature from 230° to 250°, it being necessary to carry it above the boiling point.

All piping was 6-in., with no constricted turns or openings in the system. No attempt was made to recover any light products.

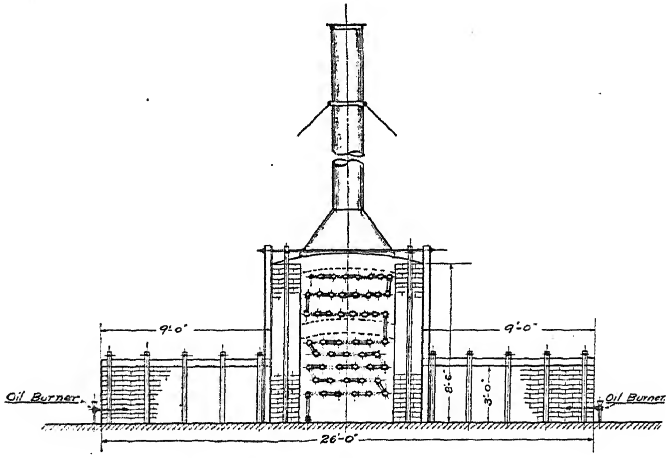

FIG. 8.—OIL-DEHYDRATING PLANT. END ELEVATION.

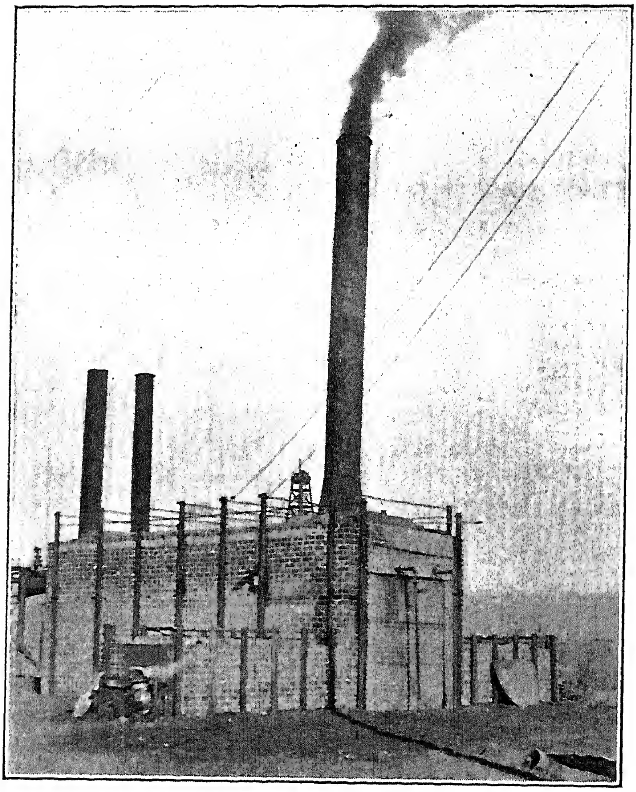

FIG. 9.—OIL-DEHYDRATING PLANT IN OPERATION.

While this plant did clean the oil, it was not a commercial success because of the excessive repairs, the sediment from the emulsion settling in the bottom of the first boiler causing blistering within a few days. These blisters had eventually to be cut out and patched and were the cause of several nasty fires owing to the boiler bagging and cracking and the oil running into the fire box. A burned-out boiler is shown in Fig. 5.

The experiment had now reached such a stage that certain definite conclusions were possible, viz:

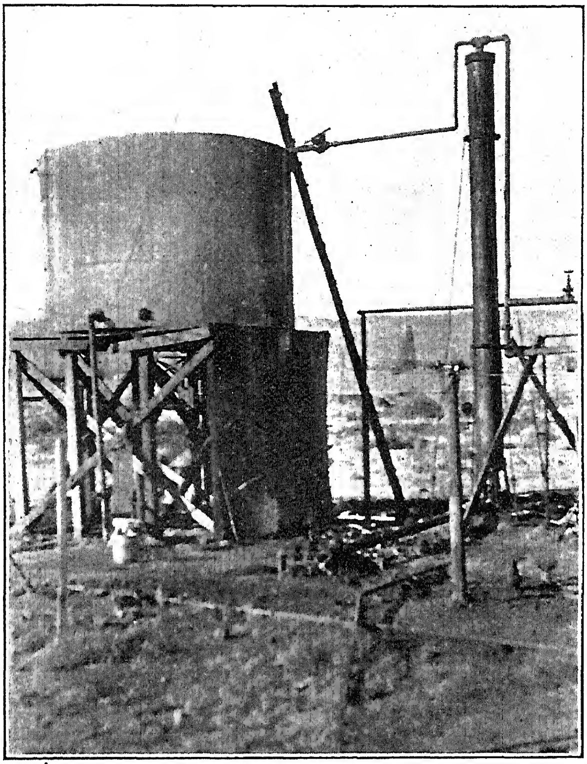

Fig. 10.—Separator and Condensing Apparatus.

1. Exhaust-steam coils cannot be used successfully for heating the oil because the temperature is not raised sufficiently high.
2. If a separation of the oil and water is to take place, the temperature must be raised to a point that will generate steam and cause an explosion in the oil-enveloped globules of water, thereby liberating the water and permitting it to separate from the oil.

The third plant was then built, consisting of brick walls lined with fire brick inside of which were coils of 3-in. pipe through which the oil was pumped. These coils were constructed in horizontal rows of five

and six 20-ft. joints below the arch and eight joints in each of the three rows above, 52 joints in all.

Furnaces were built on either side of the front so that the flames would not come in direct contact with the coil. The heat from the furnace passes into the lower compartment, thence to the back and under the arch, then through the opening at the back, into the top compartment and out through the stack at the front end.

A receiver was erected for the oil as it left the roaster, the function of which is to separate the vapor from the oil and to recover the light products that would otherwise pass off into the atmosphere and be wasted. The oil passes out at the bottom of this receiver and into the shipping tanks, the vapor passes out of the top through a coil submerged in cold water, where it is condensed and from which it is allowed to collect in a receiving tank.

This last plant, illustrated in Figs. 6 to 10, has proved a complete success. Because the coils are small, the solids from the oil cannot precipitate and form blisters. It has been operated daily for the past year without any repairs.

The cost of operating the dehydrating plant per 100 bbl. treated is as follows: Labor, 75c.; fuel consumed, 74c.; steam consumed, 22c.; total, $1.71. The condensation per 100 bbl. is: Water, 10.40; light oil, 0.43; total, 10.83. The gravity of the light oil is 37.2. The oil before treating showed: Temperature, 80° F.; specific gravity, 14.8; B. S. and M., 11.6 per cent. The oil after treatment analyzed: Temperature, 260° F., specific gravity, 15.5; B. S. and M., 1.0 per cent.

Comparative Costs of Rotary and Standard Drilling

BY M. L. REQUA, SAN FRANCISCO, CAL.

(New York Meeting, February, 1915)

In the fall of 1910, the Nevada Petroleum Co., operating in the Coalinga field in California, determined to drill a number of wells with rotary tools, in order to prove conclusively the relative value of the rotary as compared with the standard rig.

At that time, the rotary was but little known in California and its proposed introduction met with considerable criticism on the part of the operators and quiet opposition among the standard drilling crews. There were few men available who were competent to handle a rotary outfit, and these few had had but little experience in California fields. Machines as then built were much inferior to those now used, being lighter in construction and of a poorer quality of material. The shells encountered at depths between 1,700 and 2,000 ft. seemed to be too hard to permit of successful drilling, and at these depths the rig was changed over to standard tools and by them completed.

Because of the heavy depreciation, the time lost in converting from rotary to standard, and the comparatively small profit, it was concluded that unless in the future there was some material improvement in rotary rigs, nothing would be gained by drilling with rotary tools upon this property. Little or no drilling has been done upon the property since that time, but in the meantime a large number of men have been trained in the use of the rotary; in fact, many standard-tool men have abandoned standard drilling and, starting in as roustabouts, have become thoroughly competent rotary operators. The improvements in the machinery have been such as to remove many of the objections and the rotary drilling of to-day is in every way superior to that of 1910.

In contrast, a well drilled recently by the Kern Trading & Oil Co., in the east side Coalinga field may be cited. The first 2,500 ft. was drilled in 24 days and it is confidently expected that the water string will be set with a rotary at 3,400 ft.—a record obviously far beyond that made in 1910 by the Nevada Petroleum Co.

The records herewith are published with the hope that some one having more recent data will publish them so that comparison may be

Table I.—*Comparison of Cost of Drilling by Rotary and Standard Methods*

Note.—Common costs (as derrick, engines, boilers) not included.

Rotary

	2A 30	3A 30	5A 30	6A 30	7A 30	1A 30	8B 30	4 18	5 18	5A 18	Average
	$	$	$	$	$	$	$	$	$		$
Setting up rotary	0.143	0.183	0.217	0.194	0.141	0.135	0.186	0.160	0.212	0.292	0.186
Tearing down rotary	0.023	0.047	0.023	0.026	0.049	0.054	0.046	0 027	0.053	0.029	0 037
Stores to rotary	0.224	0.230	0.397	0.374	0.325	0.343	0.203	0.095	0.177	0.333	0.270
Extra materials on rotary	0.109	0.096	0.095	0 108	0.100	0 111	0 095	0.109	0 108	0 119	0.105
Extra labor on rotary	0.024	0 021	0.021	0 023	0 022	0 024	0 020	0 022	0 023	0 025	0 022
Fuel oil	0.377	0.221	0.193	0 237	0.295	0.150	0.256	0.194	0 305	0 235	0 241
Water	0.029	.....	0.047	0 029	. . .	0.046	0 037	0.070	.....	.....	0 037
Drilling labor	1.316	0 869	1.000	1 027	1.367	1 156	1.192	1.395	1.376	1.725	1.242
Standard crew casing	0.197	0 210	0.531	0 378	0 184	0.336	0 250	0.283	0.254	0.087	0.271
Rig depreciation	0 298	0 266	0 260	0 294	0.377	0 302	0.259	0.331	0.295	0 326	0.300
Casing depreciation	0 274	0.270	0 220	0 276	0 274	0.247	0 218	0.219	0.108	0.218	0.232
Overhead charges	0 242	0 102	0.079	0.211	0.205	0.094	0.170	0.081	0.157	0.194	0.153
Depreciation on tool joints, etc	0.149	0.134	0.131	0.149	0.139	0.153	0.130	0.151	0.150	0.164	0.145
Extra pump connections	0.013	0 012	0 016	0 014	0.011	0.014	0.010	0.013	0.012	0.014	0.012
	3.418	2.658	3.231	3.340	3.389	3.165	3.084	3.122	3.242	3.766	3.241
Depth, feet	1,880	2,123	2,150	1,902	2,036	1,846	2,151	1,872	1,884	1,711	1,955.5
No. of days drilling	55	62	55	56	68	54	62	62	63	65	60.3
Av. no. ft. drilled per day	32.2	34.2	39.1	33.9	29.9	34 2	34.6	30.2	29.9	26.3	32.46

Standard (1955 ft.)

	4A-30	8A-30	4A-18	Average
	$	$	$	$
Materials used	0.637	0.651	0.623	0.637
Labor	1.127	1.355	1.378	1.287
Overhead charges	0.160	0.101	0.136	0.132
Oil	0.235	0.296	0.417	0.316
Water	0.018	0.060	0.072	0.050
Depreciation charges	0.995	0.995	0.995	0.995
Cost per linear foot	3.171	3.458	3.612	3.417
Total days drilling	71	95	99	88.3
Depreciation charges made as follows:				
Bull band, calf wheel	247.50		(1/3)	82.51
Steel crown block pulley	258.75		(1/3)	86.25
Excess casing	2,561.08		60%	1,024.40
Walking beam	35.00		10%	3.50
Drill tools	3,483.75		20%	696.75
Sand line	182.00		25%	45.50
Cokely truss rod	35.00		20%	7.00
Total depreciation				1,943.00
Depreciation per foot				0.995

SUMMARY

Average standard time	= 88.3 days	Production assumed at 250	
Average rotary time	= 60.3 days	bbl. per day, 28 days	= 5,600 bbl.
Favor rotary	= 28 days	Favor rotary	
Standard average cost per ft.	= $3.42	Oil 5,600 bbl. at 50c.	= $2,800.00
Rotary average cost per ft.	= 3.24	Drilling 1,955 ft. at 18c.	= 352.00
Favor rotary	= 0.18	Total	= $3,152.00

made between 1910 and 1914. The wells drilled comprise 10 in number. The property is located about 2 miles from the railroad track and on practically level land, so that costs of hauling are a minimum. Costs would be higher upon a property at all remote from transportation, or where the ground was hilly, or the lease was not fully equipped as a going concern.

In order to compare intelligently the work done, careful costs were kept and compared with the costs of standard drilling as previously practiced upon the same property. Table I gives the detailed costs of drilling the rotary and standard wells and comparative summary. Table II shows the advance of each well for various periods of time from the commencement to the completion.

Table II.—*Depth of Wells on Various Days*

Days	2A 30	3A 30	5A 30	6A 30	7A 30	1A 30	8B 30	4 18	5A 18	5 18
10	90	......	......	......	......	......	......	......	......	100
15	540	500	100		50	400	100	170	150	450
20	800	780	400	70	200	800	300	300	400	800
25	1,050	1,150	900	350	500	1,100	600	650	670	1,050
30	1,300	1,380	1,300	750	800	1,340	800	880	950	1,280
35	1,440	1,550	1,500	1,100	1,050	1,500	1,000	1,150	1,070	1,450
40	1,550	1,700	1,680	1,450	1,280	1,700	1,280	1,400	1,180	1,580
45	1,680	1,850	1,900	1,650	1,420	1,810	1,550	1,600	1,330	1,660
50	1,780	2,010	2,050	1,780	1,590	1,830	1,750	1,690	1,530	1,710
54	......	......	......	......	......	1,846	......	......	......	......
55	1,880	......	2,150	......	1,700	......	1,950	1,760	1,650	1,800
56	......	......	......	1,902	......	......	......	......	......	......
60	......	......	......	......	1,850	......	......	......	......	......
62	......	2,123	......	......	......	......	2,151	1,872	......	......
63	......	......	......	......	2,036	......	......	......	......	1,884
65	......	......	......	......	......	......	......	......	1,711	......

Rigging up time as follows is included in above figures:

2A, 7 days	6A, 14 days	8B, 7 days
3A, 11 days	7A, 8 days	4–18, 10 days
5A, 12 days	1A, 11 days	5–18, 10 days
		5A–18, 8 days

Improved Methods of Deep Drilling in the Coalinga Oil Field, California

BY M. E. LOMBARDI, SAN FRANCISCO, CAL.

(New York Meeting, February, 1915)

THE Coalinga oil field is located on the west side of the San Joaquin Valley, California. The structure is in general a monocline, the edges of the oil horizon resting on the foot hills and dipping gently toward the east. One prominent anticline occurs plunging southeast. The earlier drilling was done in the foot hills comparatively near the outcrop and the wells were shallow. The sands were followed eastward and, in the case of the anticline, along the plunge, the wells becoming deeper and deeper until the depth of 4,000 ft. was reached and passed. There is nothing to show that the oil will not be found in quantity at still greater depth. In fact, some of the best producers have tapped the sand at close to 4,000 ft. The recovery of oil still farther to the east, and therefore at greater depth, seems to be mainly a question of drilling.

In this territory the formations drilled through are chiefly sands and shales; they will not "stand up" in an open drilling hole; the casing has to be carried close to the bit, and it is always difficult to keep the casing free for any considerable distance.

Ability to carry casing of comparatively large diameter without conductor pipes for distances of 2,000 or 3,000 ft. or over is desirable in such territory chiefly for two reasons. It makes it possible to enter the oil sand with a pipe of ample diameter; it eliminates one or more expensive strings of casing which act only as conductors for the water string, and furthermore, in territory where waters are encountered which corrode steel rapidly, it makes possible the construction of a rust- and alkali-resisting water string.

It is always desirable to shut off top waters, which may lie within 100 ft. or less of the oil sand, with 10-in. pipe. Where the depth is so great that a practical weight of 10-in. pipe will not withstand the probable collapsing pressures, 8¼ in. at least is desirable.

About the limit of rotary drilling to date in California seems to be the setting of the 10-in. string at 3,200 ft., although the rapid advance in rotary work during the past year seems to indicate that this depth may soon be increased. It is my purpose now, however, to treat only of cable-tool drilling.

The problem is to reach a depth of 4,000 ft. or more with a string of

pipe not less than 8¼-in. in diameter for shutting off top water, and to reach it with this string free and movable and using in the upper part of the hole the minimum of conductor casings.

In the Coalinga field some very promising results have recently been obtained by a method, or combination of methods, effected by William Keck. In one well a 15½-in. string was set at 2,300 ft., and a 12½-in. string through this at 3,003 ft., these being the only strings used in the well to that depth and both being landed when they were entirely free. In another well a 15½-in. string was set at 2,100 ft. and through this a 10-in. string at 3,300 ft.; in this case also the strings were perfectly free when landed and were stopped only because it was not desired to carry them deeper. Other wells have shown similar results.

It will be noted that the 15½-in. strings were free with over 2,000 ft. of "friction" on them, and this in a territory that will not "stand up" with ordinary drilling more than 40 or 50 ft. ahead of the pipe.

The drilling detail used on these wells is briefly as follows: A large clearance for the pipe is obtained; the standard circulator system is used; the pipe is kept moving while drilling is in progress; each collar is "set up" twice before it goes below the bottom of the derrick cellar.

The large clearance is obtained by the use of a shoe of extra large diameter, from 1¼ to 1¾ in. larger than the collars on the pipe. Under-reaming is resorted to frequently and the hole is repeatedly under-reamed until the pipe is entirely free in passing a "shell" or hard streak. Spudding the pipe is avoided. When a conductor pipe is landed, the desirable extra clearance for the next string may be obtained by skipping one size of pipe, as for instance carrying a 10-in. string through a 15½-in., thus eliminating the 12½-in. size. This is necessary with the present sizes of pipe available, but a different design, which will be mentioned later, would save considerable expense in this matter.

This extra clearance is necessary in using the circulator system to allow free passage for the "returns." It obviates the danger of sand lodging between the strings of pipe and freezing the working string.

The well-known standard circulator system is used. Mud- (clay) laden fluid is forced down through the pipe under pressure by pumps (ordinary rotary slush pumps) and returned on the outside of the pipe, carrying the drillings with it. This fluid is run through a flume and into a pit, as in rotary work, and its consistency is regulated as with the rotary.

This mud-laden fluid presumably plasters up the walls of the hole, prevents sand and mud from running in and prevents caving. It is essential that circulation be interrupted as little as possible. Intermittent circulation seems to be worse than useless.

The pipe is kept moving while drilling is in progress—*i.e.*, without pulling out the tools—by means of a so-called swinging spider (see Figs. 1 and 2). The pipe is suspended by an ordinary spider provided with

lugs to which are attached steel reins (sometimes chains or wire lines) which extend to a clevis above the walking beam, the beam operating between the reins. The clevis is attached to the casing block. The reins are about 40 ft. in length so that the pipe may be lowered to the bottom of a 30-ft. cellar. The cellar is made deep enough so that the stationary spider at the bottom is more than the length of one joint of pipe below the derrick floor. It follows that when a joint of pipe is added to the string the back-up tongs may be put on the second collar, which has been previously set up and which is now near the cellar bottom. The same result is obtained without back-up tongs, the pipe being held by the lower spider. Thus every joint is set up twice—once when it is put on and a second time after it has been subjected to the pull of the pipe below it and the vibration of drilling. This insures a tight joint.

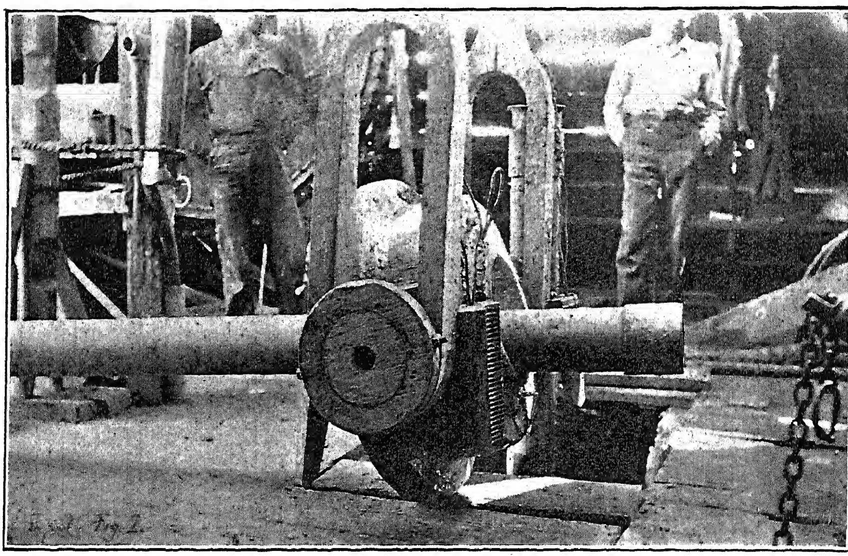

Fig. 1.—Swinging Spider.

It is by a combination of the above details and careful attention to them that success in carrying pipe has been attained.

Other advantages are obtained, one might say, as by-products of this method. The pipe is always free and the circulation perfect for cementing, the mud being easily washed out ahead of clean water. There is almost total elimination of bailing out drillings, with its consequent loss of time. A lifting pressure may be put against the closed top of the casing, thus relieving to some degree the strain on the casing line. For instance, a 12½-in. casing has an area of about 121 sq. in.; a pumping pressure of 200 lb. per inch against this means 24,200 lb. taken off the effective weight of the casing. Naturally the pressure runs up if the pipe becomes logy and that is when it is most needed.

The mud-laden fluid as usually used has a specific gravity of about 1.40; therefore, its pressure in holding back artesian water, running sand, etc., is 1.4 times as much as clear water. It is a well-known fact that this

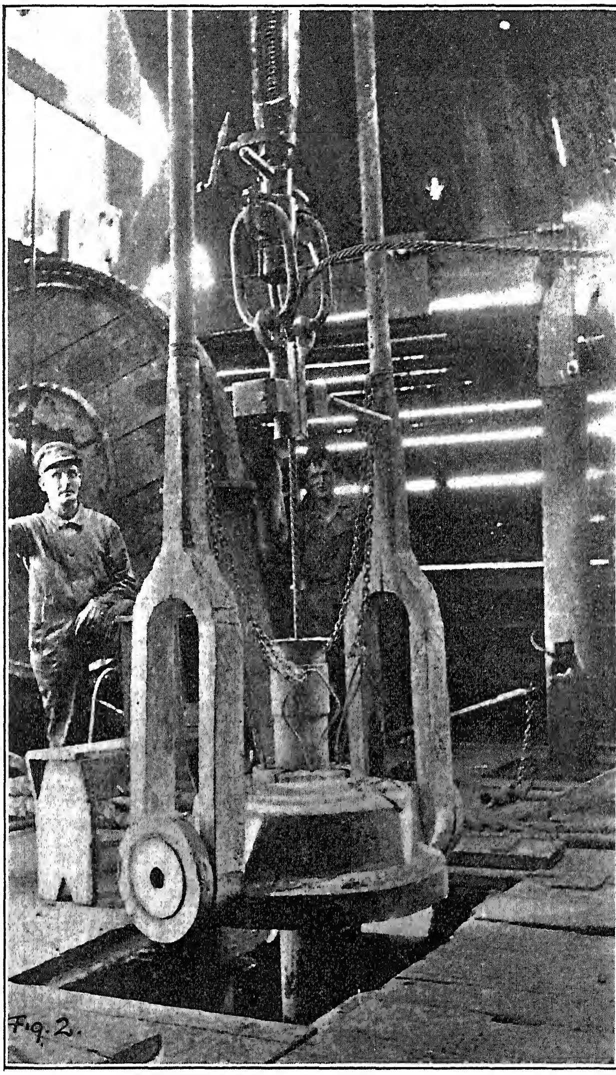

FIG. 2.—TEMPER SCREW WITH CLAMPS ON DRILLING LINE BETWEEN REINS OF DRILLING SPIDER.

mud-laden fluid tends to kill gas (see *Technical Paper No. 66, U. S. Bureau of Mines*, Mud-laden Fluid Applied to Well Drilling), although it is the writer's opinion that the capillary action of water in sand has as much to do with holding back gas pressure as anything.

It has been suggested that in territory where upper waters corrode iron and steel very fast, a 10-in. water string practically immune to this corrosion may be obtained as follows:

Carry a 12½-in. string within a few feet of the point where water is to be shut off. This string may be as light and cheap as it is practical to carry, since the burden of sustaining the collapsing pressure of the water does not fall on it. Then land a 10-in. water string inside of this 12½-in. string at the proper point below it, and pump in enough cement to fill the space between the two strings.

This is mentioned simply as one of the advantages which may accrue from a drilling method by which a large-diameter string of pipe can be carried to depth with reasonable certainty.

Now as to the interesting item of cost. The bulk of the extra cost incurred is in movable tools and machinery, only the depreciation and upkeep on which are chargeable to the well drilled. Extra cost incidental to the drilling itself, other than above, consists in construction of the deep cellar, the installation of one extra boiler, the mud pumps and the extra fuel and water used, and one extra man. The swinging spider, pumps, boiler, etc., are, of course, moved and used for successive wells.

An idea of the extra cost items may be gained from the following:

Extra depth of cellar, drain, and circulator rigging		$450.00
Setting of extra boiler and circulator pump		$250.00
Mud flume, pits, etc.		$100.00
Total fixed costs per well		$800.00
Extra labor in drilling, per day	$7.00	
Extra fuel, 7 bbl. of oil per day at 35c	2.45	
Extra water, packing, etc., per day	4.00	
Total extra costs per day for 122 days	$13.45	1,640.90
Depreciation at 18 per cent. on $2,320, being value of outfit removed.		136.00
Total extra cost		$2,576.90

In a typical well with the system under discussion 3,336 ft. of 10-in. pipe was set in 122 days.

This is the value of only about 1,465 ft. of 12½-in. 45-lb. casing f.o.b. the field. No 12½-in. casing was used, so at least this amount was saved.

Better average time is made with this method, so that at the most it is not more costly than the usual cable-tool drilling. As pointed out, its chief value lies in the fact that large-diameter pipe can be carried to depth with far more certainty.

It is impossible to leave this subject without a few suggestions for the future; in other words, indulging in a mental construction of an ideal drilling outfit, built along lines following the above described method.

A greater clearance between consecutive size strings of casing is

essential. In order to shut off water with a certain size string, the conductor used, be it long or short, should have an inside diameter at least 1¾-in. greater than the outside diameter of the collars on the water string. Since the water string will have to stand great collapsing pressure and must therefore have very thick walls, a clearance should be provided in it so that the oil string may be worked without difficulty.

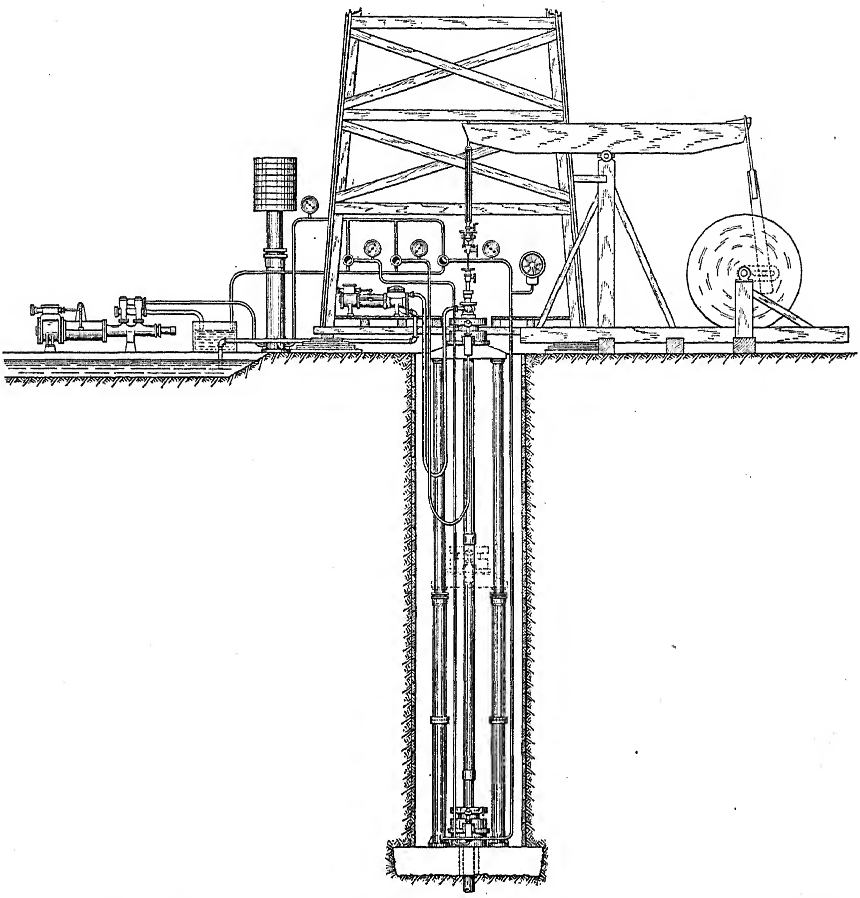

FIG. 3.—HYDRAULIC ELEVATOR FOR MOVING CASING, USED IN CONNECTION WITH CIRCULATOR SYSTEM. (PATENTED.)

If it is desired to use an 8¼-in. oil string, it follows that the water string should be at least 10⅛-in., inside diameter, which would be obtained in a casing of 10⅝-in. nominal diameter weighing from 55 to 60 lb. per foot. This would call for a 14-in. conductor.

Requirements for the ideal casing are:

Sufficient thickness of walls to withstand the water pressure; joints that will hold on a maximum pull; threads that will stand being set up several times; reduction of weight near the top of the string.

To meet these requirements the casing will have to be heavy on the bottom and light on top, but the top part will have to stand the greatest pull. Obviously an upset-end casing, upset on the outside, would be the thing for the top part of the string. Furthermore, with these upsets and with the thickness of the wall necessary on the bottom of the string, eight threads per inch could safely be used in the joints. It is well known that eight threads will stand more unscrewing and screwing up, more driving and, in general, more "grief" than the customary 10 threads.

Collars may be built correspondingly heavy since excess clearance is obtained anyhow. In explanation of the desirability of using eight threads I would say that in carrying a string of casing a long distance there are generally accidents, such as pinching a shoe, etc., which necessitate pulling and putting back the casing before it is finally landed.

The swinging spider, although efficient, is clumsy. In moving casing with it in the ordinary way, the entire strain (sometimes 80 tons or more of casing, plus friction of the mud between the casing and the walls of the hole, must be moved) is transmitted to the crown block on top of the derrick. To obviate this and carry the greatest strains on solid foundation, a hydraulic elevator operating in the cellar might be used. Such an elevator with a lift of 22 ft. has been devised, but never used to my knowledge. (See Fig. 3.) By its use pipe could be kept moving all the time, and all hard pulls taken off the crown block. Of course, the ordinary elevators and calf wheel would still have to be used for putting in and pulling strings of pipe, but the heavy work could be carried by the hydraulic elevator in the cellar, and the cost of constant repairs to the derrick considerably lightened. Pulling in of derricks, with consequent delays, frozen pipe, and danger to life, now far too frequent, would be almost entirely done away with. With the above suggested improvements, it seems reasonable to expect that an 8¼, 10, or 10⅝-in. water string, with only a few hundred feet of conductor, could be carried in the territory under discussion to 4,000 ft. or more.

Discussion

I. N. Knapp, Ardmore, Pa.—Some years ago I made a swinging spider to carry a string of casing in a way somewhat similar to that described in this paper. The ground I tried it in was too soft for its successful operation.

The spider had a central one-flange bushing to carry the slips. Under this flange and on top of the spider was a properly protected ball runway, or ball bearing, which permitted rotating the casing when it was in place and supported by the slips.

It is advantageous when a casing is moved up and down to keep it free to be able to turn it slightly, say one-quarter to one-third of a revolution, at the top of the stroke. This rotation tends to keep a better clearance for the casing than a simple up-and-down movement.

The Estimation of Oil Reserves

BY CHESTER W. WASHBURNE, NEW YORK, N. Y.

(New York Meeting, February, 1915)

At present it is impossible to estimate closely the amount of oil obtainable from a given area of land. However, after the completion of a few properly distributed prospect wells, one can calculate the approximate yield of the sands penetrated, with a probable error of say 50 per cent. Even rough predictions of this kind are of value in large operations.

Without wells upon it, one can never be absolutely certain that an area will produce any oil whatever, and one can only indulge in wild guesses of the probable productive area, before the geological structure has been carefully contoured and the top of the basal water plotted on a map. This requires wells.

At times it is desirable to form an idea of the possible yield of oil in advance of drilling, as in cases where the price asked for undeveloped land seems too high, but where the probability of finding oil is furnished by neighboring development, supported by the geology. In such cases the most one can do is to make an estimate of possible maximum yield, which may of course far exceed results, and which is of use mainly in the prevention of excessive initial investment.

The method is very significant in a general way, as in regions like the Bighorn Basin, Wyoming, where the productive sands in the Cretaceous shale are so few and so thin and fine that they cannot develop very great production, although the high quality of the oil probably will counterbalance this defect sufficiently to give local profit in favorable areas.

Reservoir Capacity of Sands.—The porosity of a stratum is the measure of its maximum reservoir capacity for liquids and gases. The porosity may be determined experimentally and must be used in connection with the total volume of the sand.

Volume of a Horizontal Sand, per Hectare and per Acre

Area	Thickness of Sand	Volume of Sand	
		in Cubic Meters	in Barrels of 42 Gal. (U. S.)
1 hectare	1 meter	10,000	62,898
1 hectare	1 foot	3,048	19,171
1 acre	1 foot	1,233	7,758

The figures of the last column multiplied by the thickness of the sand, by the porosity, and by the relative saturation, give the capacity of the sand in barrels per unit area. Thus, a sand 12 ft. thick, with a porosity of 15 per cent. and a relative saturation of 75 per cent., contains $12 \times 0.15 \times 0.75 \times 7758 = 10{,}473$ bbl. per acre. Assuming an extraction factor of 60 per cent., each acre would produce 6,284 bbl.

Under the metric system, in foreign fields, one would apply the third column, with a correction for specific gravity. Thus a sand 20 m. thick, with a porosity of 20 per cent. and a relative saturation of 60 per cent., would contain $20 \times 0.20 \times 0.60 \times 10{,}000$ or 24,000 cu. m. of oil per hectare. If the oil had a specific gravity of 0.900 this would equal $0.900 \times 24{,}000$ or 21,600 metric tons. With 60 per cent. extraction each hectare would produce 12,960 tons of oil.

Values of Factors.—The uncertainty of these figures becomes evident when one considers the doubtful factors on which they are based.

The relative saturation depends mainly on the volumes of free gas and of water which are occluded in the oil or entangled in the oil-bearing layers. It can be estimated by analysis of the product of wells in the same region that occupy analogous positions, allowance being made for the volume of gas emitted, which should be reduced to its volume under the pressure in the sand. In the absence of experiments the writer arbitrarily estimated the relative saturation at 75 and 60 per cent. in the preceding examples, the volumetric remainders of 25 and 40 per cent., respectively, being assigned to gases and water. In some fields the relative saturation decreases to zero with time, and the exhausted wells produce only water accompanied by more or less gas of some kind.

The porosity of sands varies from nearly zero to over 20 per cent., changing greatly in different parts of the same sand. Many field determinations are required to determine this value. The writer suspects that if the porosity be determined from specimens gathered on the outcrop, it will be too low, because in a recent experiment the porosity of a deeply buried sand was about one-fourth greater than the porosity of the same sand at the surface. The difference appeared to be due to the slight deposition of mineral matter (largely calcite) in the interstices of

the surface specimens. The grains of the surface specimens appeared to be identical in size, shape, etc., with those from a depth of 1,600 ft.

Moreover, underground joints and fissures may greatly increase the storage capacity of a sand. The volume of fissures in any rock and of solution cavities in limestone or dolomite is incapable of estimation. Hence the oil reserves of Mexico, a large part of which lie in cavities of this kind, cannot be estimated with anything like the approximate value that holds for Arnold's estimate of the oil reserves in the sands of California.

Extraction Factor.—The preceding figures approximate the amount of oil contained in a so-called saturated sand. Not all of this oil can be recovered under the present methods of exploitation, and the preceding result must be multiplied by a fraction representing the part of the oil which can be taken out of the ground through oil wells. This fraction, which may be called the extraction factor, is thought to vary from 60 to about 80 per cent., the higher figure being probable only in the case of gas-rich oils in sands which have been pumped to a vacuum.

The amount of oil left in the so-called exhausted pools of the Eastern States is probably a noticeable fraction of their production, as indicated by the success of the new method of forcing compressed air into wells in order to drive the remaining oil through the sand to neighboring wells, where it is pumped. The method is clearly better than that of extreme sucking with pumps after the decline of the wells, because at that stage most of the oil lies in the finer pores, as for instance in the sharper corners between the sand grains, etc., and it requires a considerable pressure gradient to move oil in such places. The vacuum method of pumping is undesirable because it does not create a high enough pressure gradient, much of the "vacuum" being overcome in the sand by the progressive vaporization of the light gasoline. This vaporization renders the little masses of oil more viscous and harder to move. The vacuum method is the only one adapted to worn-out gas sands, but it is not adapted to the requirements of old abandoned oil fields, which can best win a temporary new lease of life by the application of compressed air. In this way one can obtain all of the oily products that are susceptible of extraction, except the residual gas. It is the most economical in the end, and the most consistent with the principles of conservation.[1] It seems not unreasonable to assume that the application of the method would add 5 per cent. to the total production of a field of light oil.

The entire question is one of practical importance, and the problems involved can be solved with much greater accuracy. All we need is more data, which can be obtained through statistical studies by the U. S. Geological Survey and the U. S. Bureau of Mines.

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[1] I am told that L. Dunn of Marietta, Ohio, was the first to apply this method for the rejuvenation of old pools.

Discussion

Roswell H. Johnson, Pittsburgh, Pa.—I hope that this contribution of Mr. Washburne's is only the first of a series dealing more in detail with the very important problems involved.

Pores are of two kinds: Those which are entirely inclosed, so that they do not belong to a system of communicating pores tapped by the bore hole or shot hole; second, those which do so communicate. Porosity, of course, includes both. But since pores of the first class are valueless to us, we should confine our attention to the latter, for which I propose the term "effective porosity." This should be measured by the Wisconsin method, based on the quantity of fluid that can be introduced by the aid of vacuum.

Another factor which requires attention in this connection is the dependence of the extractability upon the rate of loss of pressure, as affected by the number of holes, and the rapidity of their completion, the existence of uncontrolled wells, the use by other operators of vacuum pumps upon the casing head, the exhaustion of a gas field in the same reservoir prior to the discovery of the oil, etc.

The extractability depends upon the pressure available for the exclusion of the oil from the reservoir into the hole. The action of gravity in bringing oil into the hole is relatively small where the pores are very small, as is so frequently the case.

Oil and Gas Possibilities of Kentucky

BY F. JULIUS FOHS, TULSA, OKLA.

(New York Meeting, February, 1915)

WITH portions of two coal basins within its borders and a few scattered fields already developed, the question arises: What is the future of Kentucky as an oil-producing State? Is the long list of failures due to lack of commercial pools, or unintelligent prospecting? A study of its beds and irregularities of structure points not only to a large waste of development money on unpromising areas, but also to the presence of a few structures well worthy of development.

The surface rocks of Kentucky show in succession more than 4,000 ft. of Paleozoic sediments and more than 2,000 ft. of Cretaceous, Tertiary, and more recent deposits. Folding and erosion have brought these beds to the surface, where they have been observed and studied in detail by my associate, James H. Gardner, myself, and others. This has given opportunity to observe the beds offering suitable reservoirs and having the proper impermeable covering; also the mapping of outcrops and outcrop lines, taken in conjunction with available well records, indicates that certain areas are worth testing, and with even greater definiteness shows areas which should be excluded, as devoid of possibilities.

West of the Tennessee River, Cretaceous, Tertiary, and Quaternary sediments occur so that all trace of ancient folding is obliterated, the sediments overlapping unconformably Mississippian rocks. This area embraces 2,000 square miles, or one-twentieth of the area of the State, the Paleozoic rocks covering the remainder. Since there is nothing upon which to base the location of tests for oil and gas in the Cretaceous-Tertiary beds of Kentucky, the Paleozoic area will be chiefly considered.

The distribution of Paleozoic rocks in Kentucky is centered about the north-northeast striking Cincinnati geanticline, bringing to the surface on the Jessamine dome the oldest rocks exposed in the State, those of the Devonian, Silurian, and Ordovician systems. On either flank of this great earth-arch are the Mississippian rocks, sloping gradually beneath the coal-measure basins to the west and to the east. West of the western coal basin, and between it and the Cretaceous-Tertiary rocks further west, is a high area of Mississippian rocks. Crossing the State

in an east-west direction is the Chestnut Ridge anticline (a disturbance recently shown by Mr. Gardner to extend from the Ozarks to the Appalachians),[1] consisting in Kentucky of the Rough Creek uplift, the Kentucky River fault zone, and the Warfield anticline. The Jessamine dome of the Cincinnati arch is due to the crossing of the Chestnut Ridge disturbance.

Beginning at the west, between the Tennessee and Tradewater rivers, lies the much-faulted area constituting a portion of the fluorspar field of western Kentucky and southern Illinois. Its structure results from the crossing of two monoclinal folds causing a fan-like fold spread with maximum dip to the southwest and the maximum uplift beyond the Tradewater River, northeast in the Rough Creek uplift of which it forms a part. Faulted with displacements exceeding 1,500 ft., and with heavy shale beds absent, presenting every opportunity for escape of oil and gas, search for them in this section would be useless. This embraces most of Crittenden, Livingston, Caldwell, Trigg, Lyon, and Christian counties. In the more southerly portion of these counties, toward the Tennessee State line, gentler dips and virtually no displacements occur, but all trace of oil structure is obscured by the red clay and chert of the Mississippian limestones, making this portion of the section one of uncertain availability.

Considering next the western coal field, we have a basin area split in an east-west direction by the Rough Creek uplift, which has a fault zone paralleling its northern slope. This uplift exposes a belt of Chester or upper Mississippian limestone and sandstone, while on its southern slope small domes occur, due to the crossing of north-south running anticlines. These domes are favorable for oil, and on one of them Mr. Gardner located the Ohio County pool. Some small exposures of oil formations have been found north of the uplift, but more favorable opportunities occur in the basin on the south. On the extreme south side of the basin the beds outcrop. The LaSalle anticline of Illinois dies out before reaching this coal basin in Kentucky. The districts in the vicinity of the asphalt-rock occurrences in Edmonson County, and of certain other localities in this basin, are favorable for further investigation, and some may warrant testing.

Locally small domes may occur on or in the vicinity of the Rough Creek uplift which would warrant testing, but such areas are few in number and restricted in size, and must be selected with great care.

On the south side of the uplift a syncline, named the Moorman by Hutchison, occurs. Its north slope is broken by faults and most of the contours on the No. 9 coal are low; probably too low to have trapped oil. On the south slope, the contours rise higher and an occasional dome or half dome occurs, usually though not always faulted. With 75 per cent.

[1] James H. Gardner; Extension of Chestnut Ridge Anticline. *Bulletin of the Geological Society of America* (1914).

of the beds consisting of shale, it is probable that many of these faults are sealed. By selection of the more favorable elevations, giving due attention to the position of the faults and their effects on the accumulation, it is possible that tests might prove the presence of a number of small pools. The south border of the coal field is a series of *en echelon* faults. Within the field and paralleling these along the line of an old fold, roughly paralleling also the main uplift, is a complex series of *en echelon* fault blocks, and testing for oil in Chester sands south of these blocks appears likely to prove of small consequence.

The Yelvington anticline, near Maceo in Daviess County, has recently been tested, but whether the hole was so placed as to be a fair test I am unable to say.

North of the Rough Creek uplift and east of the western coal field, in Meade and Breckinridge counties, minor folding and doming occur with Mississippian rocks, chiefly Chester rocks at the surface; here, in addition to small gas wells already developed, a few favorable localities are known, which will probably yield both oil and gas.

Bordering the western coal field on the south and east other gentle folds occur, notably in Logan and other counties, and this area well deserves testing. Here also Chester rocks occur at the surface. It is noticeable that in many instances, where doming occurs on the border of this coal field, Chester sands proper, when they occur at the surface, are bituminous, clear evidence of accumulation of oil prior to the erosion which has since laid them bare and permitted the evaporation ot most of the volatile constituents.

The area lying within the limits of the main Cincinnati uplift is partly faulted and lacks the necessary structural relief; in addition, the Niagaran and Corniferous sands are either eroded, or occur at very shallow depths; it offers very limited possibilities.

In and adjacent to the eastern coal field a number of undeveloped pools probably occur. In Whitley and Knox counties, immediately east of Wayne County, owing to the proximity of the Pine Mountain uplift to the east and the absence of marked secondary folds, other than the main syncline between this uplift and the crest of the Cincinnati geanticline (only 70 miles distant), good-sized pools are not to be expected. Shortly west of this main syncline are the small synclinal folds, which control the production in Wayne County, some of which remain undeveloped. Farther northeast, the axes of the Cincinnati and Pine Mountain uplifts diverge and two prominent east-northeast folds, the Sandy Hook and Warfield anticlines, appear. The igneous dikes of Elliott County occur near the Sandy Hook fold and are probably related to it. On the flanks of both folds, where terracing or doming occurs, pools may be expected, especially on the south flank of the Warfield fold in portions of Knott, Perry, Clay, Laurel, Leslie, Breathitt, Floyd and

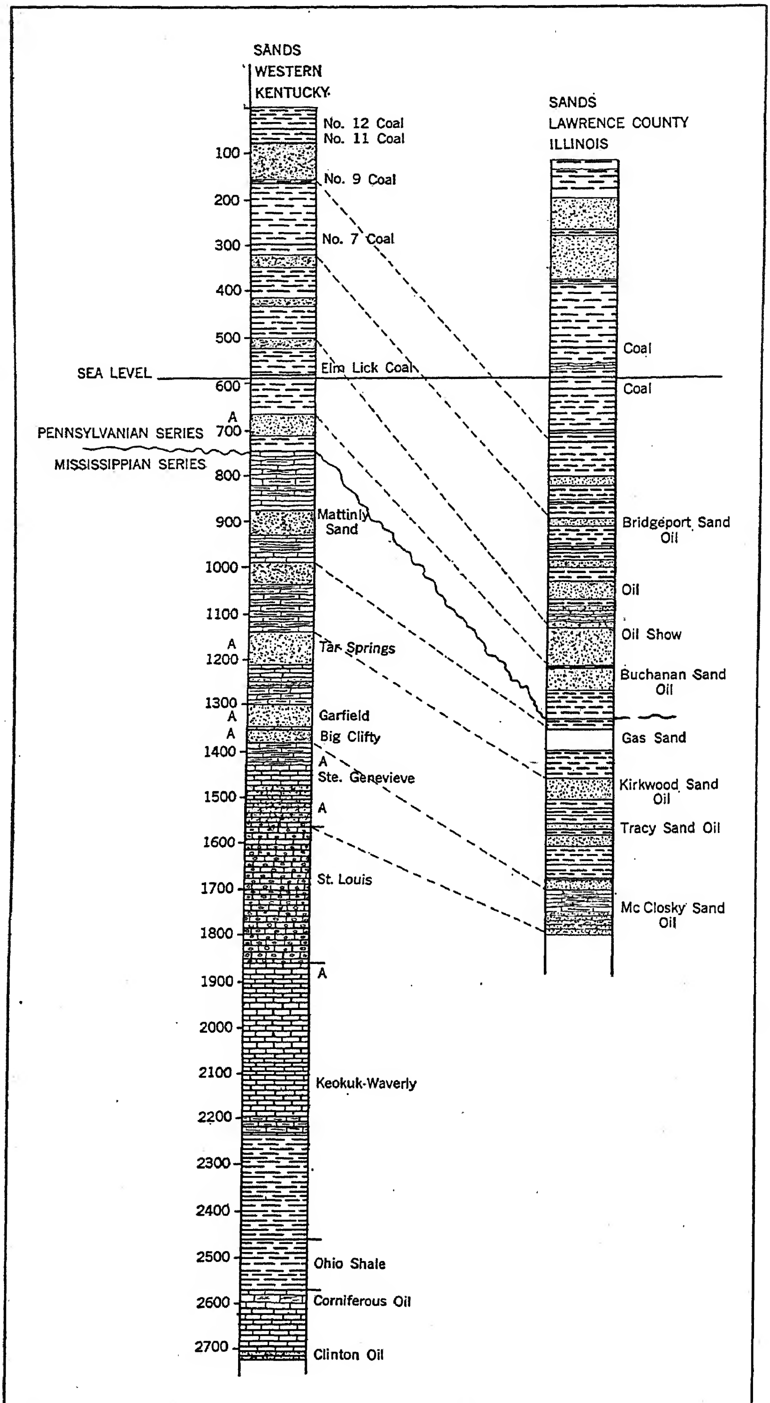

FIG. 1.—COMPARISON OF ILLINOIS AND KENTUCKY SANDS.

southern Magoffin counties. It is here that the best undeveloped areas in eastern Kentucky are to be expected.

Discussion of sands is here restricted to the five more promising districts:[2] (1) Breckinridge-Meads County; (2) interior western coal field; (3) southeastern border of western coal field, comprising western coal field and vicinity; (4) eastern flank of Cincinnati arch; (5) interior of eastern coal field, the last two comprising the eastern coal field and vicinity.

The western coal field and vicinity offer the following oil-bearing sands (see Fig. 1):

Pottsville Sands.—The Pottsville sands, represented by the Bridgeport and Buchanan sands in Illinois, give little promise in Kentucky owing to their shallow depth where the structure is favorable. The Pottsville sands, where outcropping in Edmonson County, are asphaltic.

Chester Sands.—The Chester group is represented in the Lawrence County (Illinois) field by two sands, but is here represented by five, two of which are invariably asphaltic on the outcrop, while the uppermost shows heavy salt-water saturation, which makes it promising for oil if tested under proper conditions. The Garfield and Tar Springs beds are asphaltic. The Big Clifty shows oil on structure. Where these sands are under ample cover, more than 500 ft., and of a thickness greater than 5 ft., on proper structure, they will probably prove oil bearing. These are quartz sands, and in the interior of the western coal field, at depths of 900 to 1,400 ft., should prove the best pay sands in Kentucky, but of small importance on the borders of the coal field.

Ste. Genevieve Oölites.—Where examined on the outcrop, notably on the south of this coal field near Bowling Green, these oölites carry volatile petroleum and are the equivalent of the McClosky sand of Illinois. This sand, at 1,400 to 1,600 ft. depth, should prove productive in the interior of the coal field.

Keokuk Sands.—Two productive sands are known, both dolomitic limestones, one near the top and the other 80 ft. above the Devonian or Ohio Black shale; neither of these sands has proved of commercial importance so far.

Corniferous, Niagara, and Clinton Limestones.—Each of these is 10 to 20 ft. thick and dolomitic. They offer good chances for oil and occur at depths of 1,600 to 2,000 ft., lying 30 to 200 ft. below the Ohio shale. The Carter sand of the Ohio County wells is in one of these limestones and is probably equivalent to the Boyds Creek sand (of Allen and Barren counties) of the Niagara limestone.

Cincinnatian Rocks.—In the upper beds a sandy limestone suitable for an oil reservoir may occur, but it is doubtful.

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[2] For details of logs and of sands see The Oil and Gas Sands of Kentucky, by J. B. Hoeing, *Bulletin No.* 1, *Kentucky Geological Survey* (1905).

Trenton Limestone.—In Indiana and Ohio this is one of the best productive horizons, containing 50 ft. or more of dolomitic limestone. In Kentucky, this limestone is calcitic with no dolomite beds and hence will not produce oil.

Oregon Bed.—Between the Trenton and Stones River groups, the Oregon bed, a dolomitic marble, outcrops in central Kentucky. There is nothing either for or against this bed proving productive, though on the whole we should advise against drilling deeper than the Clinton in either the eastern or western districts. The Calciferous sand would be too deep to consider drilling for in the western coal field or vicinity.

In the area west of the eastern coal field, among the oil-bearing sands are the following:

Keokuk Sand.—This is the gas sand of Wayne County, always dolomitic.

Waverly Sand.—This includes the Berea grit near the base of the Waverly. The Berea here is a fine-grained sandstone, 15 to 60 ft. thick; according to Mr. Gardner this is the Beaver sand of Wayne County.

Corniferous, Niagara, and Clinton Limestones.—These dolomitic limestones, chiefly the Corniferous, form the reservoirs in the Ragland, Campton, and other pools, while the Clinton is probably the productive sand at Cannel City. These occur at depths of 350 to 1,700 ft. The Calciferous is to be sought only on the western edge of the eastern coal field, where oil has been found in the higher sands; it occurs at a depth of 1,500 ft. below the Clinton limestone.

In the central portion of the eastern coal basin these same sands occur, but at greater depths, and in addition the Big Injun sand occurs at the top of the Waverly, while the Pottsville or conglomerate group contains the Jones, Epperson, Beaver, and Salt sands or their equivalents, with only slight chances for oil at the base of the coal measures. The Keokuk sands either are too thin or are absent here. The Chester, well represented by sands in western Kentucky, contains here either only a thin and probably valueless sand, or none at all.

The dolomitic limestones of eastern Kentucky, though rarely more than 15 to 35 ft. thick, offer wells of small size but long life; it is from these that this section of Kentucky must get the bulk of its production, though the Calciferous will later doubtlessly yield oil and gas.

In Kentucky, as in other oil-producing States, the narrow oil belts parallel the more sharply folded and faulted areas, while as distance from the axis of the large uplift is gained the folds become more and more gentle, with possibilities for larger and broader oil fields. Thus the Barren-Allen County oil fields, representing only narrow streaks of oil, give place farther west to broad and more promising terraces, as yet untested, on the southeastern edge of the western coal field; within the latter, also, exist favorable areas of which only a few have been mapped.

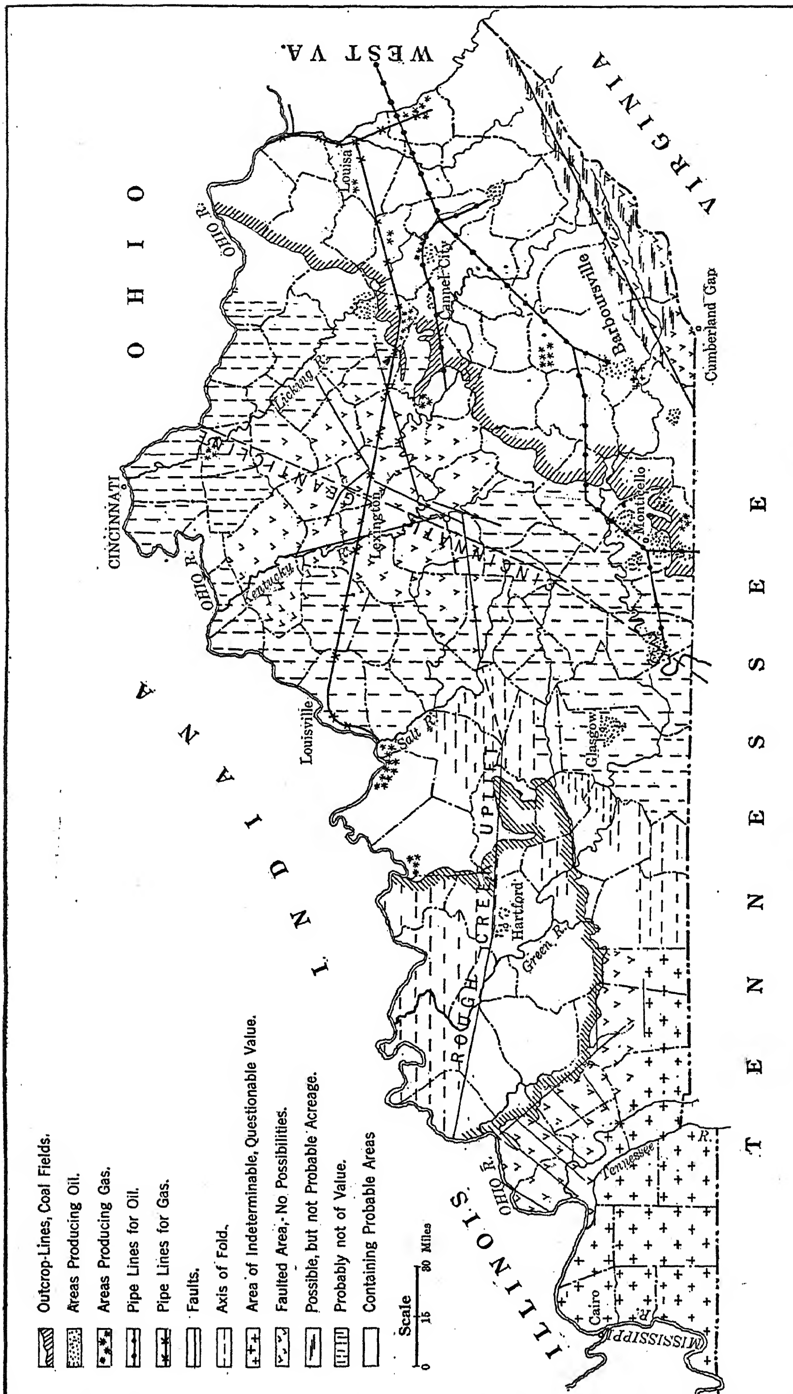

Fig. 2.—Map Showing Oil and Gas Fields of Kentucky.

It will be seen from the accompanying map (Fig. 2) that one-third of the area of Kentucky, that portion chiefly within and adjacent to the coal fields, offers chances for oil pools. Some portions of this area are already condemned and it is unlikely in any event that after careful geological investigation more than one-hundredth part of this acreage will prove worthy of testing. In addition, other areas may show possibilities, while in some no adequate surface indications exist to show whether or not they are worthy of testing. It is needless to test the faulted areas, shown on the map, which have not ample shale to seal the faults.

Of the oil pools already developed, those of the Wayne County field have yielded the bulk of production; Ragland, Barbourville, Beaver Creek, Campton, Cannel City, Hartford, Diamond Springs, and Glasgow furnish smaller but steady yields. The oil, with the exception of the Ragland field, which because of its shallow depth is heavy and of low grade, is all of the green-oil type, some even being of the amber type, and all except the first having a gravity of 35° Bé. or higher. The largest wells have produced from the Corniferous and Silurian sands, the initial flow in a few wells at Hartford and Cannel City reaching 200 and 600 bbl. In the Wayne County pools, the Beaver sand occasionally reaches an initial output of 100 bbl. shot. Of the gas pools, the only ones producing commercially in eastern Kentucky are the Menifee, Martin, Floyd, Knox (Barboursville), and (Cannel City) Morgan County fields; while in western Kentucky small gas pools are developed in the West Point, Brandenburg, and Cloverport region of Meade, Breckinridge, and Hardin counties, near Central City in Muhlenberg County and at Diamond Springs in Logan County.

That Kentucky will upon proper development yield considerable oil there is little doubt. Since most of the pools are of small size, wildcatting without careful geologic mapping in the possible areas would lead to heavy losses. Mr. Gardner and the writer mapped a few undeveloped favorable structures giving promise of pools of fair size. Owing to the prevalence of dolomitic sands of limited thickness, wells of more than 500 bbl. initial flow are apt to be rare, while most wells are likely to yield less than 100 bbl. More favorable results, if any, are, however, to be expected in the Chester sands of the interior of the western coal field. With most of its oil of high grade, Kentucky offers undeveloped fields of small daily output per well, but with good staying qualities, attractive to the oil producer rather than to the oil speculator.

Gasoline from "Synthetic" Crude Oil

BY WALTER O. SNELLING, PITTSBURGH, PA.

(New York Meeting, February, 1915)

In the course of some experiments more than five years ago, made for a totally different purpose than the investigation of the oil used, I placed a small quantity of a transparent yellow lubricating oil in a bomb-like vessel and heated it to a relatively high temperature. At the end of the experiment I removed the oil from the vessel and was amazed to find that instead of bearing any resemblance to the oil which I had put in, it now had the appearance of ordinary crude oil. The green color by reflected light and the rich red-brown by transmitted light were so unmistakable as to at once lead to further investigation. I subjected the material to fractional distillation, and the surprise which I experienced at the *appearance* of the oil, changed to amazement when I found that it yielded, on distillation, 15 per cent. of gasoline and 30 per cent. of burning oil, and that its constitution resembled crude oil quite as much as did its appearance. Furthermore, the gasoline and kerosene distillates which it yielded were of a clear water-white color, entirely without treatment with acid or alkali, and were entirely free from the odor familiar in "cracked" petroleum distillates.

The result of this experiment was quite too remarkable to be credited without further confirmation, and I at once filled the vessel with some of the same oil that I had used before, and again heated to about the same temperature that I had previously used, and for the same period of time. Upon opening the vessel and removing the contents I found, not the material resembling crude oil that I had obtained before, but apparently only the same oil that I had put in, somewhat darkened in color, but nevertheless far different in appearance from the material obtained in the previous experiment.

Evidently some condition existed in the first experiment that had not existed in the second test, and here began a series of tests in which I sought by the change of one variable after another to arrive at the identical conditions which must have existed in the first experiment. Only the fact that the bottle of heavy oil used in the first test was still in its place,

and the further fact that I had no crude oil among the materials at hand when I began the experiment—only these facts kept me from believing that I had indeed made some mistake, and that crude oil had in some manner found access to my apparatus.

After many fruitless experiments I learned a fact which should have been obvious to me from the first, but which, in the surprise due to the unlooked-for result obtained, had quite passed out of my mind. In my first test, the vessel which I used had contained but a little oil (about one-fourth of the volume of the vessel only), and in all of the other experiments I had filled the vessel three-fourths full or more, in the effort to obtain as much of a yield as possible.

I repeated the first experiment, using the vessel but one-fourth full, and heating to about the same temperature, and for the same time as I had done in the other experiments. The result was once more the greenish liquid so familiar to any one who has lived in the oil fields, and its fractionation again gave 15 per cent. of gasoline, 30 per cent. of burning oil, etc.

Apparently some remarkable change must come about in the hydrocarbon molecules, when a hydrocarbon body is heated in a still approximately only one-fourth full of oil, that does not occur when the same hydrocarbon is heated under similar conditions, except that a greater proportion of the volume of the still or retort is filled with oil. With grave doubts and fears, I placed in my retort some kerosene. If this water-white material, after treatment, should come out green in color by reflected light, and red by transmitted light, then indeed I would be convinced that I was dealing with a true transformation into crude oil. The experiment ended, I poured out from the vessel a liquid which resembled Pennsylvania crude oil so perfectly that when I placed a bottle of the new product by the side of a bottle of the real crude, it was hardly possible to say which was which, by appearance alone. I next melted some paraffin and placed it in the vessel, and after heating under the prescribed conditions, I poured out a thin fluid, suggesting crude oil in every way, and which on distillation gave somewhat over 15 per cent. of a water-white gasoline, free from "cracked" odor, and other distillates in about the same relationship as in ordinary crude oil.

One after another I tried putting all natural hydrocarbons available to me through this process. Vaseline, rod wax, gas oil, fuel oil, and B. S.—all these went into my treating vessel, one after the other. They all yielded materials similar in appearance, odor, and composition. From any of these materials I obtained a synthetic crude oil containing about 15 per cent. of gasoline, and other distillates in about the same order as are found in typical crude oils.

After many experiments had shown me the exact conditions of temperature, pressure, and filling volume of my treating vessel which were necessary to success, I fondly imagined that my troubles were over. I

did not for a moment think that human nature would involve greater difficulties than had even the control of natural conditions. Full of enthusiasm, I described the results of my experiments to an oil man, without of course describing the exact process, on which I had not yet applied for patent. He listened to me carefully and kindly, but his look of utter unbelief and incredulity was a trifle galling, to one whose life work had been devoted to scientific investigations. Had I been a promoter, selling stock in a gold mine located at Hackensack, or in a diamond mine on the outskirts of Brooklyn, I could hardly have met with less encouragement, or more entire disbelief.

To-day, when processes for increasing the yield of gasoline are being worked on by many investigators and when such lines of work are being encouraged and lavishly supported by a number of oil companies, and are being paid for in many cases with sums far greater than any probable returns, it may be hard for you to realize that only five years ago the shortest cut to suspicion and doubt, from your friends in the oil business, was even to suggest gently in ordinary conversation that perhaps by some method it might be possible to increase materially the ordinary yield of high-grade gasoline from crude oil. I tried it a few times, picking out the most kindly and genial of my friends in the oil-refining line. They would look, first pityingly and then suspiciously, and would say: "But after you have gotten out the gasoline that is present in crude oil, how do you think that you are going to get any more? Don't you understand that when you have gotten it *all* out why you have gotten it all? What is left then is kerosene, gas oil, or what-not. But it is not gasoline."

Only once did I venture mildly to protest. I suggested that possibly, since hydrocarbons were all compounds of hydrogen and carbon, it might be possible to rearrange the atoms in the molecule so as to obtain more gasoline. This view met with some recognition, and I was told that what I was talking about was called "cracking" and that it was thoroughly understood by oil men, and that, furthermore, "there was nothing in it" as far as making anything salable as gasoline, as the product would invariably be of bad color, and of an extremely offensive odor.

Slowly I came to realize that the oil industry was not yet ready for any new views as wholly different from the preconceived ideas as these experiments made necessary. So I would go back, for comfort, to the water-white gasoline of 70° Bé. which I had made from paraffin and from kerosene, from gas oil and from B. S., from fuel oil and from rod wax, and patiently wait for the day when my friends in the oil business would realize that there were a few insignificant things about oil which they did not yet know. For their attitude I could hardly blame them, after all, when I remembered my own doubt when I had seen the results of my first experiments. They had not seen them, and therefore if they doubted, I could at least understand their position, and I am hardly prepared to say that I should have been less doubting than they were, had the positions been reversed.

This paper makes public for the first time the results of my experiments, and in presenting it I wish to express my indebtedness to John T. Milliken, of St. Louis, Mo., President of the Milliken Refining Co. He was the first oil man whom I met who was willing to believe that research could really add materially to the oil man's knowledge. He has generously supported the experiments which I am now reporting, and has supplied the financial help which alone has made this paper possible to-day.

It has long been known that under the influence of high temperatures hydrocarbon bodies could be thermolized, or "cracked," and that by this method low-boiling bodies could be produced from hydrocarbons of higher gravity. Indeed, the commercial use of cracking distillation in petroleum refining goes back more than half a century, the first observation of such cracking distillation being said to have been made accidentally, in a refinery at Newark, N. J., during the winter of 1861–2.

For 30 years after the discovery of methods suitable to cracking distillation, this method was in common use in many of the principal oil refineries of the world. It was found that by running the stills at a high temperature very considerably increased yields of kerosene could be obtained, and the method was found to be a profitable one from the start, particularly since in the early days of the oil-refining industry, burning oil or kerosene was the principal product sought for. The process was also investigated by many able men, and the conditions under which long paraffin chains became broken into two or more shorter chains, under the influence of high temperatures, were very carefully worked out, particularly as concerns commercial refining operations.

Accordingly it was only natural that, with the enormous increase in demand for gasoline during the past 10 or 15 years, many investigators should attempt by similar cracking methods to obtain increased yields of low-boiling products. When the vapor of kerosene or any heavier oil is passed through a red-hot tube, for example, thermolization takes place, with the production of considerable amounts of low-boiling products vaporizing within the ordinary boiling-point range of common gasoline. In this and in many other similar ways attempts have been made, both on a laboratory scale and in large-size commercial installations, to prepare products capable of replacing gasoline. Of the dozens, or even hundreds, of such efforts, few have had even the slightest promise of success, due to the fact that the low-boiling hydrocarbons produced in the manner described are off-color, and possess an odor so pronounced and disagreeable as to greatly limit, if not wholly prevent, their sale. So acute has the demand for gasoline been at times in the past 10 years that it is not impossible that even the color and odor might have been overlooked if the process had given the large yields that were originally hoped for; but in this respect also the ordinary cracking methods have met with difficulties, and in general all these processes produce considerable amounts of tar and coke, that materially cut down their efficiency.

When the limitations of simple cracking of hydrocarbon oils at ordinary pressures were first understood, efforts were made to bring about destructive distillation under increased pressure. Results showing great improvement over those obtained by the simple cracking methods were given by these processes, which seem to have been first made use of by J. Young, and later developed by Dewar and Redwood, and others. Quite recently, improved processes of cracking distillation under increased pressures have been used commercially by Burton, and are said to have been so developed as to yield products readily salable as substitutes for gasoline.

Efforts have not been wanting to improve the color and odor of the light cracked distillates produced by ordinary cracking distillation. Treatment with sulphuric acid and alkali, in the manner commonly used in the refining of kerosene, has the effect of improving both color and odor to a remarkable extent, and by the use of sufficient acid colorless products without bad odor can be obtained, but only by the use of such large amounts of acid as to make the process commercially prohibitive, unless gasoline is selling at quite a high figure. By cracking under increased pressure the amount of acid required for this purification is very greatly reduced, and it is probably due to this fact that the motor gasoline now being so extensively developed by Burton owes its greatest commercial possibilities.

It will thus be seen that I cannot claim to be in any way a pioneer in the production of lighter hydrocarbons from materials of heavier gravity. Hydrocarbons have been cracked and broken up into lighter hydrocarbons of lower boiling point, both experimentally and commercially, for a period of over 50 years, and such cracking experiments have been conducted both at normal pressures and under increased pressures.

Other investigators have also placed hydrocarbon oils within closed vessels and have heated these oils under such conditions to elevated temperatures. In such work Engler, in particular, has made notable contributions to our knowledge of the behavior of hydrocarbons under high temperature and pressure. In these experiments it has been noted that the hydrocarbons have been broken down to lighter hydrocarbons, and that in this way low-boiling oils could be made from hydrocarbons of high boiling point. Apparently, however, the remarkable influence which is played by the ratio of the liquid contents of the vessel to the total volume of the vessel, has been either wholly overlooked or at least not properly appreciated. It has been wholly through the investigation of the effects of the ratio of the volume of oil to the total volume of the vessel that I have developed the process which I am here describing, and which has given the remarkable and unexpected results already mentioned. I believe it is only when these suitable volume relationships are observed that we can get these results within a range of temperature and pressure adapted to commercial development.

Very careful studies made in my laboratory have now proved that when a hydrocarbon body such as gas oil, for example, is heated in a vessel which is filled to more than one-tenth of its volume with such oil, but such filling is less than one-half of the total volume of such vessel, and if then the vessel is so heated that a pressure of say 800 lb. per square inch exists within the vessel, a very remarkable and fundamental change occurs in the hydrocarbon filling such vessel. It is as though the carbon and hydrogen atoms were free to rearrange themselves, and that such rearrangement goes on until a more or less definite mixture of hydrocarbons remains in the vessel. When the vessel is less than one-tenth filled with oil considerable "cracking" seems to take place and the product is quite inferior. When the vessel is much more than one-half filled with oil the reaction seems to fail almost wholly, the amount of light products produced being very small. But when the conditions within the vessel, as to amount of filling, and temperature applied, are as indicated above, the carbon and hydrogen atoms of the hydrocarbon seem to rearrange themselves to form crude oil and natural gas.

In this rearrangement, not only are low-boiling compounds produced from those of higher boiling point, but even the reverse action takes place. In several tests I have obtained from petroleum products of medium boiling point synthetic crude oils which contained high-boiling ends, whose boiling point was considerably higher than that of any of the constituents present in the original oil used. Apparently the entire process depends upon certain equilibrium reactions, in which constituents of different boiling point tend to be present in a certain very definite ratio, provided the space relationship within the treating vessel is of the proper order. Solid paraffin of course contains no constituents that are liquid or gaseous at ordinary temperatures, but upon treatment by this process even this solid paraffin is resolved into synthetic crude oil and natural gas, and the percentage of products of each definite boiling point appear to be in a definite condition of equilibrium. If instead of starting with paraffin we go to the other extreme, and start with kerosene which is entirely free from heavy ends, we will obtain a synthetic crude oil which is much lighter in gravity than that produced from paraffin, but which nevertheless contains high-boiling constituents whose boiling point exceeds by many degrees the boiling point of the heaviest product present in the untreated kerosene. Thus it will be seen that while this process is primarily one in which heavy hydrocarbons give crude oils containing light distillates (this being the main trend of the reaction), yet the process is so essentially one dependent upon equilibrium that if high-boiling constituents are absent, or present in very small amount, the equilibrium will not be satisfied until additional amounts of these high-boiling constituents have been produced as the result of the reaction which is going on.

A residual pressure, after cooling, always exists, due to the natural gas

formed in the process, and the amount of this natural gas, like the amount of gasoline in the synthetic crude oil, seems to be very constant no matter what hydrocarbon is taken. It is of course evident to the chemist that natural gas and gasoline contain a greater percentage of hydrogen than do heavier oils, and it is very interesting to note that when the charge which is placed within my treating vessel contains a hydrocarbon deficient in hydrogen, the formation of saturated gasoline goes on just the same, and the synthetic crude oil produced carries a "mud" consisting of the carbon which in the rearrangement has failed to find hydrogen. The gasoline produced from materials even highly deficient in hydrogen is quite normal in color, and does not appear to be in any way like the "cracked" products which are produced by the thermolysis of oil vapors, etc.

The following results of runs made by this process, in one case starting with solid paraffin wax, and in the other case with Oklahoma gas oil, will clearly illustrate all the technical features of the method:

Test 1.—Material used, solid white paraffin. Melting point, approximately 120° F. (50° C.). Specific gravity, 0.925 (21.5 Bé.); 300 cc. taken. Capacity of treating vessel used, 1,100 cc. Heated until pressure of 800 lb. was indicated, then cooled. Pressure of residual natural gas, 130 lb. Product after treatment, a heavy liquid, resembling "Franklin Heavy" Pennsylvania crude oil. Color, dark green by reflected light, deep red-brown by transmitted light. Volume of synthetic crude oil obtained as result of run, 305 cc. (5 cc. increase in volume over the amount of liquid paraffin started with). Specific gravity of this synthetic crude oil, 0.770 (51.8 Bé.). Gasoline yield, on distilling this synthetic crude oil to 150° C., 48 cc. Gasoline in synthetic crude oil, 16 per cent. Specific gravity of this gasoline, 0.70 (70 Bé.). Color, water-white.

Test 2.—Material used, Oklahoma gas oil. Specific gravity, 0.850 (34.5 Bé.); 300 cc. taken. Capacity of treating vessel used, 1,100 cc. Heated until pressure of 800 lb. was indicated, then cooled. Pressure of residual natural gas, 120 lb. Product after treatment, a liquid resembling Pennsylvania mixed pipe-line crude. Color, dark green by reflected light, deep red-brown by transmitted light. Volume of synthetic crude oil obtained as result of run, 288 cc. Specific gravity of this synthetic crude oil, 0.831 (38.5 Bé.). Gasoline yield on distilling this synthetic crude oil to 150° C., 40.8 cc. Specific gravity of this gasoline, 0.705 (68.5 Bé.). Gasoline in the synthetic crude oil, 13.6 per cent. Color, water-white.

It is of course evident that if putting any hydrocarbon through the process described makes it into a crude oil, it ought to be possible to take any hydrocarbon and first convert it into crude oil by the process described, then remove the gasoline, for example, or any other constituent, from this crude oil by distillation, and then to subject the residue to a repetition of the process. I have done this many times, and have converted paraffin and other petroleum products almost wholly into gasoline

and natural gas. I have obtained from paraffin about 70 per cent. of water-white gasoline, the remaining 30 per cent. representing the natural gas formed by the repeated action of the process, and some free carbon. From fuel oil, gas oil, vaseline, and similar materials, I have obtained from 50 to 70 per cent. of water-white gasoline, and samples of this gasoline, even after standing for a year or two, do not discolor, nor acquire an offensive or "cracked" odor. I wish to particularly note that this gasoline has not been treated in any way, and has never come in contact with either acid, alkali, fuller's earth, bone black, or other related materials. In brief, the process which I have described produced, from practically any hydrocarbon, a material which resembles natural crude oil, and which gives a gasoline which appears equal in quality and appearance to gasoline produced from natural crude. Both the crude oil produced by my process and the gasoline produced from its distillation possess an odor which is somewhat different from the odor of natural crude oil and ordinary gasoline. This odor, while peculiar and distinctive, is not in the slightest like the odor of "cracked" products, and it is in fact a slightly milder and sweeter odor than that of ordinary oil products. Upon mixing my synthetic crude oil, or the gasoline produced from it, with certain muds and clays, it seems to be altered, and the odor changes and becomes much more like that due to ordinary crude oil. Personally I am of the belief that crude oil in nature has in some cases been produced by some process related to that which I have here described, the effect of the high temperature which I use for a short time having in earth history been produced by very much lower temperatures acting through geological ages. I believe the condition which in my retort is represented by about three-fourths open space, in nature has had its equivalent in the open space in the sand or other porous rock which has been the repository of the oil, and I believe that natural gas, which is so commonly associated with petroleum deposits, has had a related origin in nature to that which it has in the process worked out in my laboratory experiments.

The study of the genesis of petroleum is so involved that I do not wish these suggestions to be taken in any way other than as ideas which have forced themselves on my mind after noting the very considerable similarity in appearance and constitution which exists in most of the petroleums of the world (except where a porous cover or other well-recognized conditions have allowed the more volatile materials to vaporize, or well-known oxidation or other phenomenon to take place), and it seems more than likely to me that any process which in the laboratory will produce materials of such similar appearance and composition from raw products of the most diverse nature, must surely have some connection with the conditions which in geological time have similarly produced, from starting-out products of many different kinds, a material possessing such well-marked and easily recognized characteristics as petroleum.

One very interesting development in connection with this work has

been the effect of small amounts of certain catalytic materials in facilitating the transformation into synthetic crude oil. The addition, to the oil to be treated, of even a very minute amount of colloidal graphite reduces materially the temperature and pressure at which the process is operative. In one set of experiments, in which a given treating vessel gave synthetic crude oil at an average treating pressure of 850 lb., it was found that the addition of a small amount of finely divided graphite would lower the necessary treating pressure to 750 lb., or even to 700 lb., on somewhat longer treatment. This seems to offer confirmation of the theory which I have advanced, that the entire process is dependent on certain reversible reactions which under the described conditions reach an equilibrium when sufficient time is given. The action of finely divided catalytic materials in increasing the speed of reactions is well known, and in these experiments their function seems largely to be the increasing of the rapidity of the equilibrium reactions, so that the reaction goes on more nearly to completion in the very brief time of the test. In my experiments our general procedure has been to heat the treating vessel until the desired pressure is indicated, when the heating is at once stopped, and the treating vessel cooled and emptied. We have found that when, instead of raising the pressure to the desired treating maximum, and instantly cooling the vessel, we raise to a somewhat lower temperature, and maintain this temperature for 5 or 10 min., we get practically an equivalent result. Where a catalyst is used, as described, it is possible to use a much lower pressure and still obtain a normal synthetic crude oil.

While the process has been described in this paper as an intermittent operation, the underlying principles being much more easily understood in this form, it is of course evident that the process is capable of continuous operation, and in many of my tests I have made use of apparatus adapted to such continuous transformation of heavy hydrocarbons into "synthetic" crude oil. In these experiments the heavy oil taken is forced continuously into the treating chamber, which is maintained at a suitable working temperature, and the synthetic crude oil and natural gas are continuously removed, under such conditions as maintain within the treating tube the proper relation in regard to charging volume. The synthetic crude oil produced in this way may be distilled in the usual manner, and its gasoline content removed and the residue re-treated, or by comparatively slight modifications of the process, the synthetic crude oil may be fractionated within the treating vessel, so that light hydrocarbons alone reach the discharge tube of the apparatus.

These experiments which I have described have been wholly of a laboratory nature, and much work remains to be done in the application of the principles which have been discovered to commercial work on a large scale. While it may seem to many that the pressures and temperatures employed are so high as to preclude the possibilities of commercial work, yet I do not think this is the case. Processes have been developed abroad,

during the past few years, in which ammonia is made synthetically by reactions requiring both higher pressures and higher temperatures than those which are made use of in my present work. As these ammonia researches have gone on from their laboratory inception to their commercial development upon a very extensive and successful scale, I believe the present process will find similar development comparatively easy. The conditions necessary for successful commercial work are already well known, and involve no engineering features which American ingenuity cannot easily provide, and it is my hope that this process will soon be developed to the point where it will fulfill commercially the remarkable promise that it now seems to offer.

Discussion

A. F. Lucas, Washington, D. C.—Are the results of the experiments in practical operation?

Walter O. Snelling.—There have been no large-scale operations up to the present. The necessary conditions of temperature are easily arranged for, but the required condition of pressure, 800 lb., is not so simple a matter. The company has been working on this problem, and at the refinery in Oklahoma work of this kind is now being carried on. The way in which an ordinary still will leak at 50 lb., and more at 100 lb., leaves some doubt in the minds of many as to what can be done at a continuous working pressure of 800 lb. On the other hand, the fact that they are making ammonia abroad by a method which employs higher pressures and temperatures than anything involved in this method, leads me to believe that the difficulties, which will undoubtedly arise when this is attempted on a commercial scale, can be overcome.

William N. Best, New York, N. Y.—Is there any change in the calorific value of these oils?

Walter O. Snelling.—I can only answer that in this way—in one of our experiments we took 300 cc. of paraffin, and the product which came from it amounted to 305 cc., an actual increase in volume, which is what one, as a rule, expects, when a material that is heavy is changed into a substance of lighter gravity. The number of heat units present in the original 300 cc. of paraffin were all present in the final 305 cc. of synthetic crude oil, and the heat units were in that proportion in the final product. The gasoline produced varies in gravity. This is determined by where we cut it off. We always aim at 70° Bé. The tests of the gasoline with engines, and in burning, indicate that it is apparently quite similar to ordinary gasoline, but wholly different from the cracked product produced by passing oil vapors through tubes at ordinary pressures. It is similar to, if not absolutely identical with, the hydrocarbons normal in the gasoline from ordinary crude oil.

F. G. Cottrell, San Francisco, Cal.—What temperature do you use?

Walter O. Snelling.—The criterion we generally use is pressure, for the reason that in laboratory experiments the temperature is more difficult to determine in the interior of a vessel of this kind, but of course the temperature corresponds to the vapor pressure produced.

M. R. Wolfard, Cambridge, Mass.—Was any method used to exhaust the air, and has the air above the oil any particular influence on the process? I think it would be interesting if we had definite determinations of the heating value of the oil. It is rather peculiar you can get from 100 cc. of oil 105 cc. of crude oil which normally has a higher heating value than heavier oil. It would seem necessary, therefore, to make calorimetric tests in order to demonstrate the fact positively that there is no change in the heating value. It would also be interesting to know whether synthetic gasoline always has precisely the same heating value although the density of the original oil may be different.

Walter O. Snelling.—In regard to the first matter, I may say that one of my assistants has used natural gas and other gases in the remaining part of the vessel not filled with oil, and we have also made tests with that space evacuated, but we could not see that it made any great difference, except that the exclusion of the air led to better results, for the reason that, as our criterion was pressure, naturally if there was no original pressure in our vessel, the temperature used was somewhat higher.

In continuous processes, the gases occupying the space over the oil play a very significant part, and their composition is a matter of considerable importance. Fortunately, however, this is a matter which adjusts itself very nicely, and since the gases over the oil are a portion of the equilibrium system existing, any change in the conditions of equilibrium leads to a corresponding change in the composition of the gases present.

As to the second point, it is a pretty well recognized law of thermodynamics and conservation of energy, that with a given mass of material placed in a vessel, and nothing going out, and with all the material which was placed in the vessel still in it when you get through, there will be no change in total energy. If we start with 300 cc. of paraffin, when we get through, we have 305 cc. of the synthetic crude oil. The gravity of this synthetic crude oil is lighter than the paraffin which went in. The number of heat units in the synthetic crude oil can scarcely be considered to be different from the number of heat units which went in, because according to the conservation of matter, you have a certain number of carbon atoms and hydrogen atoms, and it is the burning of this carbon and hydrogen which gives the heat.

David T. Day, Washington, D. C.—This matter is very interesting. It is evidently in a very unfinished state yet, as Mr. Snelling admits, but I think we should be very grateful to him for bringing this preliminary evidence before us so promptly and so helpfully. While there is a great deal of work going on on this subject in many different lines, there are many interesting results, and these results must be classed among them.

There are several small matters in regard to which I would like more information. I am deeply interested in the fact that certain products were obtained here under certain conditions; for example, that if you had the bomb almost full of yellow lubricating oil, say three-quarters full, and then heated it so as to produce a pressure of 800 lb. to the square inch, you would not change it from that yellow oil. This is the most remarkable statement that has been made. We have this fact, that the proportion of liquid, to begin with, in the bottle must be three-elevenths to get the maximum effect, and if you have it three-quarters full you get no effect whatever. That is extremely remarkable, and I would doubt whether continued experiment would always bring that same result, because the temperature of 300° or 250° and 800 lb. pressure of that oil in any kind of vessel is going to change the oil, I think, and perhaps that result was more or less accidental, particularly in view of the fact that you got no change whatever under these other conditions. As long as you can change oils when you heat them and compress them in this way in tubes and in bottles, etc., in so many different ways, and the fact that the odors of these oils are not those of crude oils, I fail to see the importance of the generalization that Mr. Snelling is inclined to make that this has some relation to the crude oils in the earth. I doubt whether the natural light-colored oils have been changed in nature into dark-colored oils by such a process as this, as would be inferred from the generalization made by Mr. Snelling. I fail to see the force of that. I do not think that Calgary gasoline has been changed into heavier oils where we find them in Canada.

If after heating an oil under pressure this way, and obtaining so much light oils, the light oils are distilled off, and the residue then treated the same way, and that process is repeated several times until, say, 70 per cent. of the oil had been converted into gasoline, what percentage of coke was obtained?

Walter O. Snelling.—Dr. Day's question brings up several interesting points I did not take time to speak of, because I feared to trespass upon your patience. In the first place, in regard to the contention that it is not possible under these conditions to heat an oil and have it remain but little changed: If we take, for example, some gasoline, or any light oil, you can put it through this process and it will come out wholly unchanged, and you will be able to see the reason why, since the criterion

is not temperature at all, but pressure. A pressure of 800 lb. can be produced by the heating, and the gasoline be wholly unaltered chemically. In my earlier workings with liquefied natural gas, we used a nominal pressure of 200, 300, or even 500 lb., in bottles of this gas. I have taken these bottles and placed them in warm water in order to test the containers, under the Interstate Commerce Commission specifications, and even after a pressure of 2,000 or 3,000 lb. per square inch was reached, the gasoline was wholly unchanged. The low-boiling material, heated to a point at which it was quite volatile, produced by its volatility the high pressure registered. No chemical change occurred in such case, as can be readily seen.

DAVID T. DAY.—What temperature, 300° C.?

WALTER O. SNELLING.—The temperature in our experiments with the conversion of hydrocarbons is such as produces within the vessel a pressure of about 800 lb. per square inch.

DAVID T. DAY.—Do I understand that a yellow lubricating oil, an oil like that, has ever been heated in a bomb to 800 lb. pressure and to a temperature of 250° to 300° C., and remained like that? The temperature feature is essential to the discussion I have made.

WALTER O. SNELLING.—As I have not a complete set of my experiments, here I can answer that in this way—that we have, for the purposes of our own records, made many experiments, using pyrometers in the tubes of these vessels. We have the correlation reasonably exact between temperature and pressure. There is always, when oil is heated, more or less of a change, and that change will result in causing a production of tarry products, etc. That yellow oil, when it occupies three-quarters of the volume of the treating vessel, may be heated to a high temperature, without producing material which resembles crude oil or contains any perceptible amount of uncracked gasoline. If that oil be placed in a vessel and that vessel be three-quarters full, and be brought up to the temperature necessary to bring about cracking, then of course it will be altered, and it will undergo changes which will cause the color to change, but there will be no considerable production of gasoline.

LEONARD WALDO, New York, N. Y.—Does the temperature enter into this reaction?

WALTER O. SNELLING.—In many of our experiments, the habit of my assistant has been to bring the vessel up and watch the gauge, and when it reaches 800 or 900, simply to take it off, and push the vessel under a stream of cold water. We made some experiments in which we continued the heat. We found a little different result. It is, apparently, a reversible reaction. As soon as the conditions are favorable for this change to go on, it goes on, until it can no longer go on; in other words, it goes

on to equilibrium. This change apparently goes on until a sufficient amount of material, crude oil, is produced, so that each one of its components has a certain partial pressure. You can see how that must be—as soon as a certain amount of gasoline is produced, that temperature alone causes this gasoline to have a certain back pressure. If the vessel were then to remain over night at that temperature, I feel sure, from the results of our experiments, there would not be much further change, and when the material came out it would be quite the same In chemistry in the last few years we have become familiar with catalytic action and have learned that certain reactions which take place do so under certain given conditions. We also learn that any reaction which will take place at all can have its reaction velocity facilitated or quickened by the presence of certain catalysis, and almost any reaction which will take place with the catalytic mass in quick time will take place in itself if a sufficient length of time is given; in other words, the same set of conditions that produce a material in an hour or in a minute in the laboratory, could, at a far lower temperature, provided the same fundamental condition or underlying condition existed, produce the same effect in a longer period of time. We are familiar with sets of experiments in which a certain thing goes on to equilibrium with catalysis in 5 min. at a high temperature, and that same thing would take weeks, months, or years to occur at low temperatures, and therefore one significance of this, as shown by these synthetic petroleums, is their similarity to natural crude.

A. F. Lucas.—Can you connect any relationship in your experiments with the origin of oil?

Walter O. Snelling.—Only to this extent, that if things so diverse in characteristics as kerosene, rod wax, fuel oil, or paraffin, when put under this given set of conditions give always a product which resembles one thing, natural crude oil, then it appears to me not improbable that this similarity may have significance, in view of the fact that we have reasons to believe that petroleum may have been formed in the earth from many different materials. Let me take a somewhat analogous case, in which our society has played such a remarkably prominent part. Not many years ago, the origin of ore deposits was a subject but little understood. One set of men, who were familiar with the ore deposits in certain mining districts, insisted that ore deposits had been formed by uprising mineralized waters, coming from the depths of the earth. Other men contended that ore deposits were formed by the leaching and segregation of the minute amounts of heavy metals present in the adjacent rocks. The two magnificent volumes which this society has published on ore deposits and which form unquestionably the most important contributions which have ever been made to our knowledge of the subject, show us rather conclusively that both these sets of men were wholly right. Some ore deposits have been formed by mineralized waters, from great depths,

and some mineral deposits have been formed by the leaching and redeposition of minerals, from the very slightly mineralized rock near the surface of the earth.

To-day there are many prominent oil men who believe that petroleum has come from certain transformations of vegetable or animal matter within the sedimentary rocks of the earth's surface. Others are contending that petroleum has a deep-seated source, and that this source is quite likely related in some manner to the barysphere of the earth. It is my belief that the coming years will prove, just as has happened already in the case of ore deposits, that all of these gentlemen are right. The petroleum in certain known deposits has unquestionably been formed by certain changes which have come about in accumulations of vegetable matter. Other petroleum deposits unquestionably owe their origin to the animal remains present within certain shales with which these oils are associated. Still other deposits of petroleum, I believe, have had a deep-seated source, and have originated through the chemical transformation of gases and vapors coming from comparatively great depths. Since the studies which I have outlined in this paper show that materials of such different appearance as fuel oil, kerosene, and paraffin may all under perfectly definite conditions of temperature and pressure change into a material resembling crude oil so closely that even an expert may be deceived, then at once we can see why all the generalizations which have been drawn in regard to the color, gasoline content, bloom, etc., of oils, as proving something about their origin, are wholly illusory. Starting from the position that practically all hydrocarbons may, under certain well-defined conditions, produce a material which we may consider to be crude oil, it seems to me that we stand in a favorable position to review our existing knowledge in regard to petroleum deposits, and to weigh in a fair and unprejudiced manner the arguments which all the parties have brought forward, favoring either a vegetable, a mineral, or an animal origin for petroleum. Personally I believe that petroleum can be, and in earth history has been, produced from all three kingdoms, mineral, animal, and vegetable.

Roswell H. Johnson, Pittsburgh, Pa.—From a geological standpoint, David White's recent work, which was given before the Geological Society of America in December, seems particularly interesting in the light of this discussion. He found that the quality of the oil varied with the type of organic compounds in the accompanying shales—that in Pennsylvania as we moved toward the district where we got our harder coals, those with higher fixed carbon proportions, that there also the organic matter in shales was higher in fixed carbon, and that the petroleums varied step by step also. I do not venture to say what the interrelation of these two things is, at this time, but all these hydrocarbon transformation experiments must now take on a new interest to the geologist.

Gold-Bearing Gravels of Beauce County, Quebec.

BY J. B. TYRRELL, TORONTO, ONT., CANADA.

(New York Meeting, February, 1915)

A SHORT time ago I paid a visit to the alluvial gold fields on the tributaries of the Chaudière River in Beauce County, Quebec, in company with A. O. Dufresne, late manager of the Champs d'Or Rigaud-Veaudreuil, and now assistant to the Superintendent of Mines of the Province of Quebec. As the conditions under which the gold occurs in this district are not very generally known, and present some interesting features, a brief description of these conditions, and a consideration of the causes which gave rise to them, may be of interest to other mining engineers.

During the latter half of last century the country was visited by many mining engineers and geologists, and many references to it may be found in the reports of the Geological Survey of Canada between 1848 and 1911. The most important of these are by J. A. Dresser and J. Keele and the late Robert Chalmers. The Department of Colonization and Mines of the Province of Quebec also published a report with map on the district by J. Obalski.

From the earliest times the valley of the Chaudière River formed one of the main avenues of approach to the St. Lawrence in the vicinity of Quebec from the country to the south as far as the seaboard of the States of Maine and Massachusetts. The Indians had a well-known trail along the banks of the stream, and armed troops and foraging parties constantly moved backward and forward along it between Quebec and New England in those insecure times before the ceding of Canada to Great Britain.

Even yet the natural advantages of this old military route are recognized, and the government of the Province of Quebec has now under construction along it a magnificent highway from Quebec to the International Boundary Line, which is to form part of a through highway from that city to Boston.

The land along the valley is naturally well suited for agriculture, and about the end of the eighteenth and the beginning of the nineteenth centuries farmers began to go back from the St. Lawrence, and to occupy the higher lands along the upper portions of the Chaudière Valley. This settlement has gone on until now the country is peopled with a prosperous agricultural population.

Discovery of Gold

In 1823 or 1824, a woman first discovered gold in the Chaudière Valley near the mouth of Gilbert River. No attention was paid to the discovery, but in 1834 a young girl named Clothilde Gilbert, taking a horse to water, found in the creek, close to the location of the previous discovery, a nugget of gold weighing 44 dwt. Eleven years later the DeLery family, owners of the seigniory of Rigaud-Vaudreuil, obtained a patent from the Crown giving them exclusive privileges forever to mine the precious metals within their seigniory.

In 1847, the year before gold was discovered in California, the Chaudière Mining Co., which leased the mining rights from Mr. DeLery, mined gold on the Gilbert and Des Plantes rivers, and during the three following years continued to operate on the Gilbert River.

In 1851, the mining rights of the whole seigniory were leased to Dr. James Douglas and others of Quebec, who continued operations, chiefly on the Gilbert River, until 1864. After this date mining was prosecuted with more or less activity for about 30 years.

In all, up to the end of the century, about $2,000,000 worth of gold was extracted from the gravels of the Gilbert River valley, while it would seem that about $500,000 worth of gold was extracted from the gravel of the other tributaries of the Chaudière River.

Character of Country

That portion of the watershed of the Chaudière River and its tributaries, from whose buried gravels gold to the value of $2,500,000 has been extracted, extends for 20 miles in the direction of the valley, and 6 miles transverse to it, forming a block of land about 120 square miles in area, in which placer mining has been more or less systematically prosecuted. It lies in Beauce County, Quebec, 50 miles southeast of the city of Quebec, and 25 miles west of the International Boundary Line between Quebec and the State of Maine. The principal town is Beauceville, with 1,700 inhabitants, situated on both banks of the river at an elevation of 500 ft. above the sea, with hills rising to heights of 600 or 700 ft. both to the northeast and southwest of it. Transportation to or from the district is afforded by the Quebec Central Railway, which at the present time runs two passenger trains a day each way to and from Quebec. The railway runs up the valley of the Chaudière River through a number of small prosperous towns which are located on the bank of the stream, while back from the river the country is laid out in farms which are for the most part cleared of timber and in a good state of cultivation. Two wagon roads run up the valley, one on each side of the stream, and the method which has been generally adopted here, as elsewhere in Quebec, of surveying farms with a

narrow frontage on the river and a long extension back from it, permits the farmers to live moderately close to one another beside the main roads, giving these roads the appearance of long-extended scattered villages.

The country in which the gold-bearing district is situated is a dissected plain or tableland with a mean elevation of 1,000 or 1,100 ft. above the sea, lying between two old and greatly degraded mountain ranges.

These mountains are the northern extensions of the Green Mountains of Vermont and the White Mountains of New Hampshire. They run in two parallel chains about 50 miles apart northeastward from the International Boundary into the Gaspé Peninsula. The stronger chain, which has been called the Megantic Range, runs along the International Boundary Line, and some of its peaks rise to heights of 2,500 or 3,000 ft. above the sea. Some of the peaks of the other chain, known as the Sutton Range, rise as high as those farther to the southeast, but taken as a whole this range is the lower of the two.

Between these two ranges of mountains lies an extensive tableland which has been worn down by long-continued atmospheric erosion into rounded hills and wide valleys. The summits of the hills are covered with a thin mantle of glacial drift, while the lower slopes are rounded up by a thicker layer of the same unassorted material. In their native condition the hills were completely covered by magnificent forests of pine and maple, now largely cut down since the land has been brought under cultivation.

Drainage

The general direction of the drainage from this tableland is either northeastward or southwestward, parallel to the mountains. Nevertheless, it is trenched across, and the Sutton Mountains are cut through by the great transverse valley of the Chaudière, which collects the water from the many normal longitudinal streams, and carries it down into the St. Lawrence River. This valley has been cut deep into the old plateau and has reached a fairly mature condition, with gentle slopes descending from the high lands on both sides to the river, which has a moderate and fairly regular grade of about 8 ft. to the mile from the upper portion of the area under consideration to the St. Lawrence River. Such minor obstructions as do occur in the stream, as at the Devils' Rapids, have probably been caused by diversion of the river from its old channel by glacial agencies.

Structural Geology

The rocks that compose the Sutton and Megantic mountains are pre-Cambrian gneisses, and talcose, chloritic, and micaceous schists.

Between these mountain ranges, in the region of the Chaudière, the

plateau country is underlain by green and reddish slates, quartzites, and sandstones, which are stated by the officers of the Geological Survey of Canada to be of Cambrian and Cambro-Silurian age. In many cases these slates, etc., present a remarkable similarity to the pre-Cambrian slates and schists of Keewatin age in northern and western Ontario. Some of the slates are ordinary water-worn sediments, while others have recently been proved to be ash rocks, or similar rocks of igneous origin.

These rocks were deposited in a horizontal attitude in the seas of the Paleozoic era, but have been squeezed and crushed so that they are now generally steeply inclined or even vertical, and strike about N. 45° E., parallel with the mountain ranges.

Through the schists and slates, dikes and bosses of igneous rock, varying in character from peridotite to quartz-porphyry, have been injected. It is highly probable that some of the igneous rocks intercalated with the slates were injected into them as sills or laccolites before they were tilted and folded into their present attitude, but some of the dikes are doubtless subsequent to the folding. However, it is significant of the age of the igneous rocks associated with the gold-bearing gravels in the vicinity of Beauceville, that some of the green schists, associated with and included in the folding of the Cambro-Silurian rock in the valleys of Mill Creek and Chaudière River, were found to be volcanic rhyolite tuffs, while the igneous rocks in the vicinity are quartz-porphyries of similar composition, and probably of somewhat similar age.

Mr. Dresser[1] has already pointed out the similarity in character of these intrusives to those of the Stoke Mountain range farther west, which are associated with some of the most important copper properties in Quebec.

In the valley of Gilbert River, from which most gold has been collected, quartz-porphyries and acid intrusives, either sills or dikes, are particularly abundant. In the vicinity of many of the more acid intrusives quartz veins have been found to occur containing more or less gold associated with such sulphides as pyrite, chalcopyrite, and galena. These are all the hard rocks known to exist in the district under consideration, and such sediments as overlie them consist of unconsolidated material of very much younger age.

The oldest of the later sediments consist of thin beds of stratified gravel extending down the bottoms of the valleys, but of no considerable lateral extent. In places they contain grains and nuggets of gold. Overlying these gravels is a varying thickness, sometimes as much as 100 ft., of unassorted and unstratified boulder clay. Other and later sands and gravels also occur in gorges in the bottoms of the valleys, which also contain a small quantity of gold. Overlying these is a second thickness

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[1] J. A. Dresser: The Bedrock of the Gilbert River Gold-fields, Quebec. *Journal of the Canadian Mining Institute*, vol. viii, pp. 259 to 262 (1905).

of boulder clay. Finally there is gravel in the bottoms of the present streams.

Historical Geology

The sequence of events which led up to the formation of these buried gold-bearing gravel deposits was about as follows:

After both the igneous and sedimentary rocks of early Paleozoic and pre-Paleozoic times had been formed or deposited and had been intensely crushed and folded into what must have been a range of mountains, they appear to have been intruded by dikes of the following igneous rocks: Peridotite, pyroxenite, gabbro and diabase, granite, quartz-porphyry, etc.

Subsequent to these intrusions, probably to the last of them, the rocks were again subjected to heavy strains, so that they were still further fractured. Into some of the more acid of the igneous rocks (whether dikes or sills is not always certain), siliceous waters carrying sulphides of iron and copper, with native gold, were introduced along the fractures, also from these fractures the gold-bearing solutions seeped out into the adjoining rocks, forming quartz veins and pyritized zones carrying a smaller or larger percentage of gold. Thus the veins were formed from which the grains and nuggets of gold found in the valley gravels have undoubtedly been derived.

Toward the close of the Paleozoic era, and after the rocks had assumed a fairly stable condition, the whole country was raised above the level of the sea, and since that time it would appear to have remained above sea level, and to have been exposed constantly to the influence of atmospheric and stream erosion and denudation. During this vast period of time, extending from the end of the Paleozoic era to the present, an enormous thickness of rock was undoubtedly removed from the general surface, and as the softer rocks would be worn away faster than the harder ones, the latter remained as higher points and ridges.

At first the water which drained from the district would flow downward to the sea over the lowest parts of the surface, irrespective of the hardness of the rocks of which this surface was composed, and water courses so begun might persist to the present. The great valley of the Chaudière is probably such a persistent water course, while the smaller streams have been cut off from their direct connection with the sea, and have been obliged to become tributary to the Chaudière, their courses being finally determined by the varying characters of the underlying rock.

While the surface was being decomposed through the agencies of air and moisture, with the help of plants and animals, the decomposed rock was constantly being carried downward by the rills and streams, and at the same time was being assorted into heavier and lighter portions. In this process the coarser and heavier portions constantly lagged behind and became entrapped by the inequalities of the underlying rock, while the

smaller and lighter portions were carried down into the main channel of the Chaudière River, and thence into the sea.

In this way, during the long period which intervened between the uplift near the close of the Paleozoic era and the beginning of the Pleistocene period, the country was worn down, possibly from a high range of mountains, to a fairly mature physiographic relief, in which rocky cliffs and gorges were unknown, and the slopes of the hills were everywhere gentle, with coverings of decomposed residual rock. Also in the bottoms of the wide valleys the streams flowed with gentle regular current without rapids or waterfalls. In and beside these streams were deposits of sand and gravel which undoubtedly contained most of the heavy minerals that had been washed down from the adjoining hills during the whole period of their long-continued erosion, unless these minerals had been carried away in solution, or were in a sufficiently fine state of division to have been transported to the sea with the lighter sediments. Of these heavy minerals the most important, and at the same time the most persistent, was gold.

The general relief of the country at the beginning of the Pleistocene period would have been very much like that of the Klondike district, in Yukon Territory, at present, particularly those parts of the Klondike drained by Dominion Creek and Indian River, where later gorges have not been developed; with this difference, that the Quebec slopes were easier and the whole topography was more mature.

Another point of similarity between the two districts is, that throughout the whole time when active erosion was in progress the drainage of the country was local and the whole of the gravel concentrated in the bottom of any valley was derived from that particular valley or its tributaries, and not from a foreign valley.

Again, the gravel in the bottom of a valley was the ultimate concentrate from the vast quantity of material which had been eroded from that valley, possibly aggregating many cubic miles of rock, and consequently if the gravel was rich in gold it was due to the quantity of rock concentrated, rather than to the original high gold tenor of the rock.[2]

At the beginning of the Pleistocene period there was a break in the continuous course of atmospheric and stream erosion which had been in progress throughout the Mesozoic and Tertiary epochs, for snow and ice began to collect in great quantity on the Appalachian Mountains to the south, and from this center or gathering ground the ice moved northwestward down the valley of the Chaudière River, across the hills which flank it on both sides, and over the valleys of the tributaries which flow into it

[2] Cf. The Gold of the Klondike, by J. B. Tyrrell, *Transactions of the Royal Society of Canada*, vol. vi, New Series, Sect. 4, pp. 29 to 59 (1912), and The Law of the Paystreak in Placer Deposits, by J. B. Tyrrell, *Transactions of the Institution of Mining and Metallurgy*, vol. xxi, pp. 593 to 613 (1912).

approximately at right angles to its course, until it stopped in the vicinity of the south bank of the St. Lawrence River.[3]

From the standpoint of the miner engaged in the exploitation of alluvial gold-bearing deposits, this first ice invasion from the south is of great interest, for, inasmuch as it moved down the Chaudière Valley, where this valley runs northwestward, it doubtless removed any stratified sand and gravel which may have been in the bottom of those portions of the valley so oriented, and at the same time it rounded up the sides of the valley, and filled in the mouths of the lateral valleys with débris collected from the valley itself or from the sides of the adjoining hills. During its later waning stages it probably also left lateral moraines on both sides of the valley.

When at its greatest extent, this Appalachian glacier covered the higher lands and moved over the valleys of the tributaries of the Chaudière River which were transverse to its general course. In these cases it moved the decomposed rock from the summits and the south sides of the ridges down into the valleys and covered the gravel, which had previously been deposited there, with a coating of boulder clay or till.

In most cases, as in the valley of the Gilbert River, the glacier had lost the greater part of its pushing power when it reached the lower levels, so that it left the gravels undisturbed and merely covered them with its heavy coating of dirt brought from above. In some few cases, as in some places on the banks of Meules Creek, there was still a little vertical energy left in the glacier when it reached the bottom of the valley, and so it kneaded up the sand and gravel into a compact unstratified mass of water-worn material a few feet in thickness before covering it with unassorted till.

While this northwestward moving glacier pushed a certain quantity of loose unassorted material into these smaller transverse valleys it did not fill them, but deposited its load on their southern slopes, and consequently when it retired it left the new bottoms of these valleys farther north than they were before, while the old pre-glacial gravels in the original bottoms of the valleys were buried under the talus of rock débris to the south.

When the Appalachian ice withdrew from the country at the close of the first glacial period, the brooks and rivers flowed in the same valleys which they had occupied before the ice invasion, but as the bottoms of the transverse valleys had been moved toward the northwest the streams naturally adopted the lowest parts of the valleys, and therefore now flowed in channels northwest of their former channels, and usually at somewhat

[3] For detailed evidence of the succession of events during the Glacial Period the reader is referred to Robert Chalmers's Report on the Surface Geology and Auriferous Deposits of South-Eastern Quebec, *Annual Report of the Geological Survey of Canada*, vol. x, New Series (1897).

higher elevation; at the same time they were cut off from the main Chaudière Valley by the ridges or lateral moraines which had been piled up along its sides. Consequently, in their endeavor to reach the main stream, the lateral brooks cut new gorges in the bottoms of the valleys northwest of the old channels, but their sides remained steep, for the period during which the country was free from ice does not appear to have been sufficiently long to have permitted of the grading of the sides of these second gorges to gradual slopes. One of these interglacial gorges has been outlined by shafts and drill holes on the northwest side of Meules Creek.

After the deep, narrow, interglacial gorges had been formed the country was again, and probably more deeply, covered with ice, but on this occasion the ice accumulated on the Laurentian hills north of the St. Lawrence and then moved southward and southeastward across the St. Lawrence River and up the long slope south of it for about 100 miles almost to the summit of the Megantic range of mountains on the International Boundary Line. This second invasion of ice therefore moved up

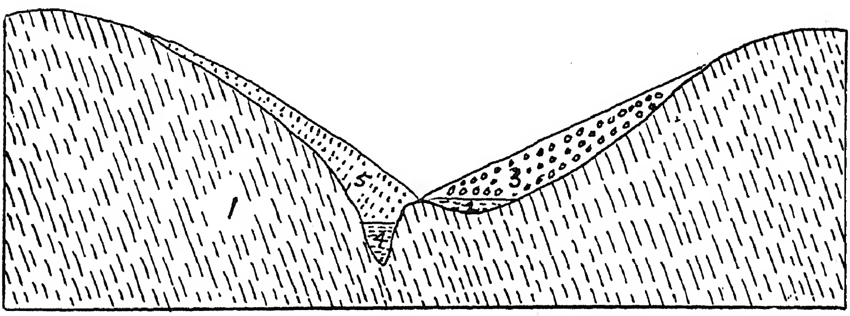

Fig. 1.—Diagrammatic Section across the Valley of Meules Creek, which flows Northeastward into the Chaudiere River.

1. Paleozoic slate.
2. Pre-glacial gold-bearing gravel.
3. Boulder clay of the Appalachian glacier from the southeast.
4. Interglacial sand and gravel.
5. Boulder clay of the Laurentian glacier from the northwest.

the valley of the Chaudière River in the opposite direction to that in which it had moved on the former occasion. Again it scored out and smoothed off the bottom and sides of the main valley. Also, as it passed over the valleys tributary to the main valley, and at right angles to its course, it pushed such decomposed and broken rock as it was able to collect down into these valleys, covering their northern sides with débris and filling in and covering up the interglacial gorges which had recently been cut in them, but it did not completely fill the valleys with boulder clay, so that when this glacier in its turn melted away and disappeared, and open streams again began to drain the country, they flowed in channels independent of either of the earlier channels, and in some cases at least, intermediate between them. (See Fig. 1.)

Since the close of the last glacial period, when the ice finally retired from the country and left it in much the same condition as it is at the present time, the streams in the transverse valleys are again cutting new channels for themselves in the bottoms of the valleys through the covering of boulder clay and down into the underlying rock on lines independent of the earlier channels.

A striking feature of the new system of drainage which prevails in the country at the present time is that the lateral streams discharge into the main valley of the Chaudière over rapids or waterfalls from "hanging valleys." This condition indicates clearly that the lower parts of these lateral streams are not now occupying their old pre-glacial or interglacial channels. In no case was I able to learn of either one or the other of the old channels having been traced all the way down into the Chaudière channel.

When the ice had finally retired it left the whole country, both hills and valleys, covered with a sheet of glacial drift. On the hills this sheet is usually thin, while in some parts of the valleys it may reach a thickness of 100 ft.

This sketch of the causes which led to the formation of the beds of gold-bearing alluvial gravels, and of the methods which Nature adopted in giving them their present characteristics, and in hiding them in their present obscure locations, may be summarized as follows:

Summary of Gold Conditions

1. Gold was probably introduced into the folded Paleozoic rocks subsequent to, but in close association with, sills or dikes of acid rocks, such as quartz or granite-porphyry.

2. It was introduced along with pyrite and other sulphides in siliceous water which formed quartz veins in or near these dikes, etc.

3. Toward the close of the Paleozoic Era the country was raised above the level of the sea, and has remained above the sea until the present time.

4. Throughout the most of this immensely long period, until the beginning of the Pleistocene period, it was constantly suffering erosion from atmospheric and stream agencies, and it was worn down to a fairly mature condition with gently sloping hills and wide valleys.

5. Where gold occurred in these hills it had been washed down through countless ages into the bottoms of the valleys, and was concentrated in the alluvial gravels beneath and beside the streams.

Many streams throughout northern Canada which flowed over gold-bearing rocks also must have had gold-bearing gravel in their beds in pre-glacial times. In most cases, however, the subsequent glaciation was sufficiently severe to have carried away all this gravel, while in the Chaudière district the glaciation was less severe, and some of the gravel was left in place.

6. After this long period of erosion and concentration a great glacier formed on the summit of the Appalachian Mountains and moved northwestward over the country toward the St. Lawrence River. On its way it crossed the valleys which lay transverse to its course, and buried some of the gravel which lay in the bottoms of those valleys under a heavy mantle of boulder clay. Sometimes the gravel was left quite undisturbed in its original condition, sometimes it was kneaded together so that its stratified character was obliterated. It is chiefly from these pre-glacial beds of gravel that gold has been extracted.

As the glacier moved directly down the Chaudière valley it probably scored out most, if not all, of the gravel which had accumulated in it, though up to the present time this question does not appear to have been definitely settled, for the bottom of the valley has not been thoroughly prospected either by shafts or drill holes. At one place, namely at Devils' Rapids, gold has been found in the Chaudière River, but here the stream is flowing for a short distance transverse to the general direction of the valley and the course of glaciation.

7. After the Appalachian glacier had retired, new and narrow channels were cut by the transverse streams in the bottoms of the transverse valleys, to the north of the old pre-glacial channels. These contain a small quantity of gold, but the interglacial period was not sufficiently long to permit of the concentration of much gold in them, so that except where they may possibly have cut into or across the earlier pre-glacial channels they have not proved, and are not likely to prove, rich in gold content.

Up to the present time interglacial channels do not appear to have been distinguished from pre-glacial ones, and doubtless some of the failures which have occurred in the district have been caused by expending time and energy on the buried, but poor, interglacial channels, under the impression that they were the rich pre-glacial channels.

8. After the interglacial channels were formed another glacier advanced across the country from the northwest and buried these channels under another and later sheet of boulder clay.

9. When this glacier retired from the country the streams began to cut out their present channels, which are independent of the two former sets, but as yet no large quantity of gold has been concentrated into these new channels.

Whether all the buried gold-bearing gravels have been discovered or not will not be discussed here, but it may be pointed out that the pre-glacial channels of the lateral streams, which in their upper courses are gold-bearing, do not appear in a single case to have been traced down to their junctions with the main valley of the Chaudière River, though it is reasonably certain, from the mature character of the topography throughout the country, that such channels are continuous without falls or interruptions from the lateral valleys into the main valley.

Character of Bedrock

The bedrock underlying the pre-glacial gold-bearing gravels consists chiefly of green and gray chloritic and quartzitic slates striking N. 45° E. and dipping southeastward at an angle of 70° or steeper. Where these slates are overlain by pre-glacial gravels they are rough and uneven and form excellent natural riffles, so that the gold was collected either in the inequalities of their surface, or immediately above them, and they have not been smoothed and polished by glacial agencies like the rocks of the adjoining hills.

On the Gilbert River and in other localities these slates were intruded by sills or dikes of quartz-porphyry, and these sills or dikes occur at places which are said to have been the most productive in the whole area; but in the absence of personal observation of the old mines it is impossible for me to say what effect the character of this bedrock had on the pay streak in the old channels.

Character of Gold

The gold obtained from the gravels of the tributaries of the Chaudière River is mostly such as is usually known to placer miners as coarse gold, very little of it being in the form of very minute flakes or particles. There can be little doubt but that fine gold existed in the veins from which the placer gold was originally derived, but if it did it was carried farther down the streams and much of it was probably deposited in the gravels of the Chaudière River. One nugget was found on the Gilbert River which weighed 51 oz., 18 dwt., 6 grains, another weighed 45 oz., 12 dwt., while another nugget found on the same river weighed 42 oz.

Last summer Louis Matthieu recovered 50 oz. of gold from the gravels of Meules Creek, all of which was quite coarse and granular. The largest nugget weighed between 2 and 3 oz., while the next largest, which I obtained, weighed 24 dwt., 12 grains.

Mr. Obalski gives the fineness of two samples of gold as 874 and 879, equal to a value of $18.06 to $18.15, and these may be considered as representing the average fineness of the gold of the district.

Methods of Mining

In the earliest days of mining in the district the gravel was collected from the bars in the river, probably where the river crossed or cut into one of the old channels, and was washed in a gold pan or in a cradle or rocker to recover the gold.

Afterward some parts of the pre-glacial channels were discovered which were covered with but thin layers of boulder clay. The boulder

clay was thrown to one side, and the underlying gravel was shoveled from the open pits into sluice boxes, supplied with water from higher up the river, and the gold was collected in the boxes.

At a later period the rich gravels were found under a heavy overburden of boulder clay, in places 60 or 70 ft. thick. In some cases this buried gravel was reached by tunnels driven into the sides of the hills, and in other cases by vertical shafts sunk to it. From the ends of the tunnels and from the bottoms of the shafts as much of the gravel and underlying bedrock as contained gold was mined and brought to the surface, where it was washed in sluice boxes as before, and the gold extracted.

Four or five years ago a much more ambitious plant was installed on Meules Creek. A ditch 7 miles long was dug from Lake Fortin, at the head of Mill Creek, in which water was conducted to a penstock on the high ground south of Meules Creek. Thence it was conducted in a pipe to a point on Meules Creek where hydraulic operations had been determined upon, the head of the water at this point being 260 ft. Here one or more hydraulic giants were installed and through them the water was projected against the south side of the valley, and the gravel and débris were washed down and run through a sluice to collect the gold. At the tail of the sluice a bucket elevator picked up the tailings and stacked them lower down the valley. Unfortunately, the operations do not appear to have been financially successful, for the bank which it was proposed to wash down proved to consist of boulder clay with but a thin layer of re-assorted pre-glacial material beneath it, which was not sufficiently rich to compensate for the poverty of the boulder clay above. The plant has not been in operation for the past two summers.

During the past summer a few tributers or laymen were working in a small way "shoveling in" on Meules Creek, with the result stated at the beginning of this paper, but mining work appears to have ceased on Gilbert, Des Plantes, and other streams in the vicinity some years ago.

A Study of the Chloridizing Roast and Its Application to the Separation of Copper from Nickel

BY BOYD DUDLEY, JR., STATE COLLEGE, PA.

(New York Meeting, February, 1915)

THE material presented in this paper is an abstract of a thesis submitted by the writer to the faculty of the Massachusetts Institute of Technology, as part requirement for the degree of Master of Science. The investigation was undertaken at the suggestion of Prof. H. O. Hofman, and it was conducted, under his direction, in the laboratories of the Mining Department of the Institute.

The purpose of the work was, first, to study some of the reactions that have been considered as a part of the mechanism of the chloridizing roast, and second, to study the conditions under which nickel oxide and copper oxide may be chloridized. While the production of chlorine gas by various reactions during the chloridizing roast may not be primarily useful in the chloridation of metal oxides and compounds, it nevertheless serves a useful purpose in preventing the dissociation and decomposition of chlorides once they have been formed; for example, the dissociation of cupric chloride into cuprous chloride and chlorine. Therefore it was thought that a study of some chlorine-producing reactions would prove of interest, and the first series of experiments dealt with this phase of the subject. The object of the latter part of the work was to show the possibility of treating heavy sulphide ores, such as those of Sudbury, Ontario, containing much iron and small amounts of copper and nickel, by chloridizing and removing the copper, leaving an iron-nickel oxide product suitable for smelting in the blast furnace for nickel-bearing pig iron. The chloridizing of the copper would, of course, be preceded by a roast for the removal of sulphur, the sulphur oxides being available for the manufacture of sulphuric acid, if under the local conditions this product would be of commercial value.

REACTIONS OF THE CHLORIDIZING ROAST

Reaction of Sodium Chloride, Sulphur Trioxide, and Oxygen.—The first reaction studied was that involving the production of chlorine

from sodium chloride by the action upon it of sulphur trioxide and atmospheric oxygen. This reaction may be expressed by the equation, $4\ NaCl + 2\ SO_3 + O_2 = 2\ Na_2SO_4 + 2\ Cl_2$. In the roasting furnace the sulphur trioxide is derived from the decomposition of sulphates that have been formed in the roasting ore, and the oxygen is supplied by the air in contact with the roast.

The action of sulphur trioxide alone upon sodium chloride has been studied by E. Kothny,[1] who concludes as follows: By treating sodium

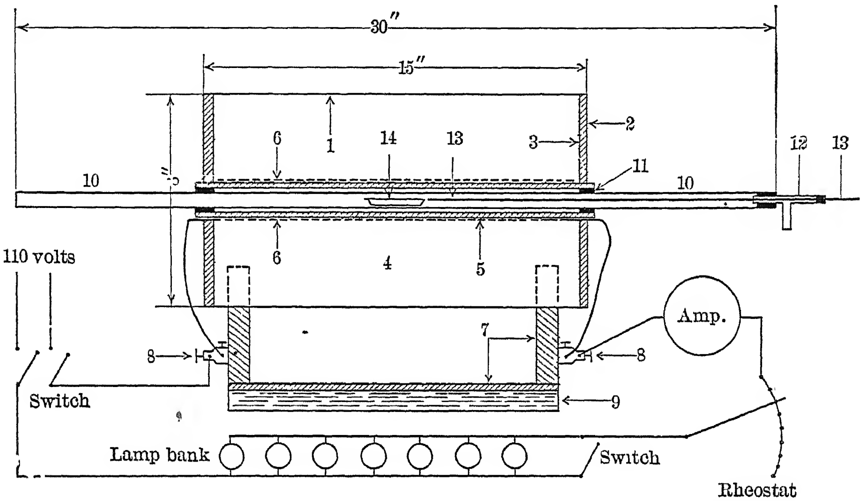

FIG. 1.—CROSS-SECTION OF ELECTRIC RESISTANCE FURNACE.

1. Cylindrical shell of galvanized iron.
2. End of shell of galvanized iron.
3. Asbestos board 1/4 in. thick.
4. Magnesia.
5. Porcelain tube about which resistance tape is wound. Internal diameter 1 in.
6. Resistance winding of "Excello" alloy tape, 5.3 mm. wide by 0.4 mm. thick.
7. Asbestos board.
8. Binding posts.
9. Wood board.
10. Glazed porcelain combustion tube. Internal diameter 0.6 in.
11. Asbestos packing.
12. Glass T through which pyrometer tube and air enter 10.
13. Silica pyrometer tube.
14. Boat containing material to be heated.

chloride with sulphur trioxide there probably results $NaClSO_3$, which at ordinary temperatures is stable only in an atmosphere of sulphur trioxide, and decomposes at 150° C. with the evolution of sulphur dioxide and chlorine. When the sulphur trioxide is diluted with air or carbon monoxide the salt decomposes at ordinary temperatures. At a red heat sulphur trioxide and sodium choride react and give sodium sulphate, sulphur dioxide, chlorine, and probably sodium pyro-sulphate.

[1] *Berg und Hüttenmännisches Jahrbuch*, vol. lviii, p. 350 (1910).

As was stated above, the experiments herein described involved the action of a mixture of sulphur trioxide and air upon sodium chloride; they therefore differ from those of Kothny, and more nearly approximate the conditions obtaining on the hearth of a roasting furnace. In these experiments it was not proved that the atmospheric oxygen actually enters into the reaction, nor was any attempt made to determine the equilibrium conditions of the reaction. It was considered sufficient to show that dry sulphur trioxide mixed with air reacts with

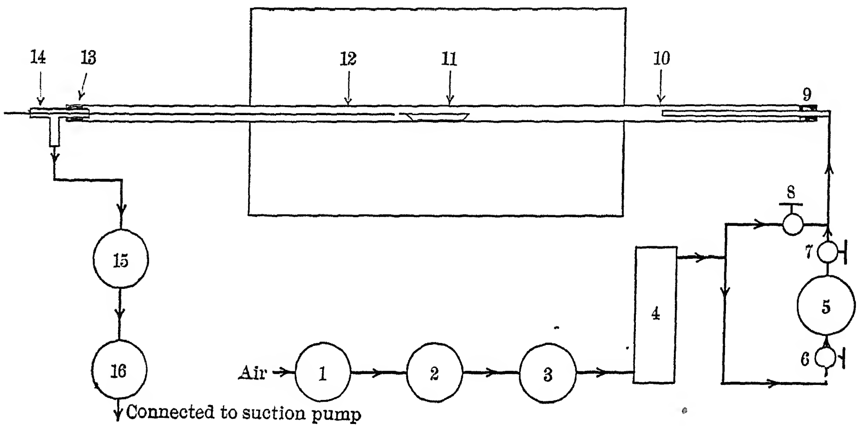

FIG. 2.—APPARATUS FOR SECURING DRY MIXTURE AND INTRODUCING IT INTO FURNACE.

1. Wash bottle containing potassium permanganate solution.
2. Wash bottle containing potassium hydroxide solution.
3. Wash bottle containing sulphuric acid.
4. Drying tower containing phosphorus pentoxide.
5. U-tube containing sulphur trioxide and phosphorus pentoxide.

6, 7, 8. Glass stopcocks.

9. Plug of litharge-glycerine cement.
10. Porcelain combustion tube.
11. Porcelain boat containing sodium chloride.
12. Silica pyrometer tube.
13. Rubber stopper.
14. Glass T through which pyrometer tube and gases pass.

15 and 16. Glass railroad tubes for absorbing chlorine.

sodium chloride to produce chlorine, and to show the effect of temperature upon the rate of reaction.

Method of Conducting the Experiments.—In the first experiments sodium chloride contained in a porcelain boat was heated in an electric tube furnace, through which was passed the mixture of sulphur trioxide and air. The rates at which the chlorine was evolved at different temperatures were determined by passing the gases through a solution of silver nitrate, and weighing the silver chloride precipitated in a known time.

Before placing the salt in the furnace it was heated to a temperature slightly below its melting point, after which the lumps formed during

the ignition were broken in an agate mortar. In this way decrepitation in the furnace, with the consequent loss of material from the boat, was avoided. The salt was not pulverized, but was left coarse in order to expose as great a surface to the action of the gases as possible.

The details of the construction of the furnace and its electrical connections are shown in Fig. 1.

It was important to secure a mixture of sulphur trioxide and air free from water, because, as a result of the presence of water vapor, hydrochloric acid would be produced, and this would mask the tests for chlorine that could be readily made under the conditions of the experiments. The method of securing the dry mixture and of introducing it into the furnace is shown in Fig. 2. The train of apparatus was

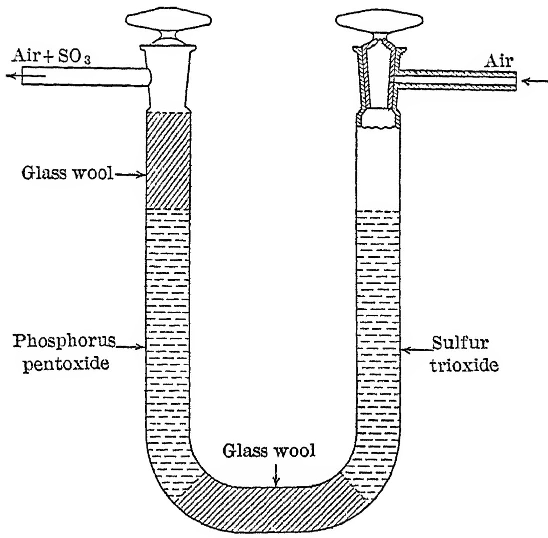

FIG. 3.—SULPHUR TRIOXIDE CONTAINER, 5 OF FIG. 2.

assembled as shown, the air being drawn through a series of wash bottles containing potassium permanganate solution, potassium hydroxide solution, and concentrated sulphuric acid, then through a 10-in. glass absorption tower containing phosphorus pentoxide, in the order mentioned. From the drying tower the air could be admitted directly into the furnace tube through the stopcock (8); in this manner the tube and its contents could be thoroughly dried. By closing (8) and opening (6) and (7) the air could be drawn through the sulphur trioxide container, which was a U-tube containing solid sulphur trioxide, phosphorus pentoxide, and glass wool arranged as shown in Fig. 3. All of the connections between the sulphur trioxide container and the furnace were made by blowing glass tubes together, thus avoiding the use of rubber or other

connectors that might be attacked by the gas. The stopcocks were lubricated with syrupy phosphoric acid. No attempt was made to regulate closely the composition of the gas mixture entering the furnace. The air passing through the U-tube, mixed with the sulphur trioxide vapor and carried it away, the concentration of the mixture depending upon the vapor tension of the sulphur trioxide and the rate of flow of the air. Since the room temperature, which governs the vapor tension of the sulphur trioxide, and the rate of flow of the air were both practically constant in all cases, it is safe to assume that the composition of the gas mixture was practically the same in all of the experiments.

Upon leaving the furnace tube the gases passed through two railroad tubes containing a solution of silver nitrate, made by dissolving 3.5 g. of the salt in 250 cc. of nitric acid and diluting to 500 cc. with distilled water. It was found that this solution would absorb chlorine, forming a precipitate of silver chloride, without allowing the sulphur trioxide and dioxide present in the furnace gases to interfere by precipitating the silver as sulphate and sulphite, respectively.

Preliminary experiments with this apparatus, conducted by introducing the boat of salt into the cold furnace and then slowly raising the temperature while the gas mixture was being passed through the tube and into the absorption tubes, showed that the evolution of chlorine began with sufficient rapidity to be detected by the indicator at about 500° C. When the temperature was less than this the indicator showed no cloudiness.

In order to determine the effect of temperature on the rate of the reaction, a number of tests were made in which the temperature was the only variable. A porcelain boat containing 1 g. of sodium chloride that had been previously ignited was inserted into the hot furnace and allowed to come to furnace temperature, a current of dry air being meanwhile passed through the tube. At the end of about 10 min. the mixture of sulphur trioxide and air was admitted to the tube and was allowed to pass for 15 min., at the end of which time dry air was again passed for 10 min. The absorption tubes were then disconnected from the furnace tube and the boat was removed. The silver chloride that had been precipitated in the absorption tubes was removed, filtered from the solution, washed, ignited, and weighed. The salt was removed from the boat, and dissolved in water acidified with hydrochloric acid. After heating the solution to boiling an excess of barium chloride solution was added, and, after standing, the resulting barium sulphate was filtered from the solution and weighed The results of this series of experiments are shown in the following table:

1	2	3	4	5	6
Temperature, Degrees C.	AgCl, Gram	$BaSO_4$, Gram	SO_3, Gram	Cl, Gram	Cl (calc.), Gram
533	trace	trace	....	....	....
597	0.0020	0.0048	0.0016	0.0005	0.0014
700	0.0136	0.0107	0.0037	0.0034	0.0032
710	0.0083	0.0087	0.0030	0.0020	0.0026
758	0.0255	0.0234	0.0080	0.0063	0.0070
769	0.0488	0.0554	0.0190	0.0137	0.0120

In the table above column No. 1 shows the temperature to which the salt was heated. No. 2 shows the amounts of silver chloride found in the absorption tube at the end of 15 min. No. 3 shows the amounts of barium sulphate obtained by the addition of barium chloride to the contents of the boat. No. 4 shows the amounts of sulphur trioxide contained in the barium sulphate precipitates. No. 5 shows the amounts of chlorine contained in the silver chloride precipitates. No. 6 shows the amounts of chlorine that correspond with the amounts of sulphur trioxide found combined with the salt, if one molecule of chlorine is liberated by the combination of one molecule of sulphur trioxide with the salt.

From the results of these experiments it appears that the rate of the reaction becomes appreciable at about 500° C. and increases rapidly with increasing temperature. The agreement between the amounts of chlorine found and those calculated from the amounts of sulphate found, while not close, is reasonably good considering the difficulties of manipulation and the small amounts of material worked with. The fact that there is no consistent variation between these quantities tends to indicate that each molecule of sulphur trioxide that combines with the salt liberates one molecule of chlorine. However, a more detailed study of the reaction would be necessary in order to fully establish this fact. From these results it also appears that the reaction is a comparatively slow one even at temperatures near the melting point of the salt; but since this apparent slowness could be attributed to a failure of the gases to come into intimate contact with the salt, it was decided to determine the effect of a more thorough contact upon the rate of chlorine evolution. In the series of experiments just described the highest temperature attained was 769° C.; the effect of higher temperature was not investigated, because it was not desired to melt the salt. At temperatures below 500° C. the indicators always remained clear, and no tests for sulphate could be obtained from the salt in the boat after the heating.

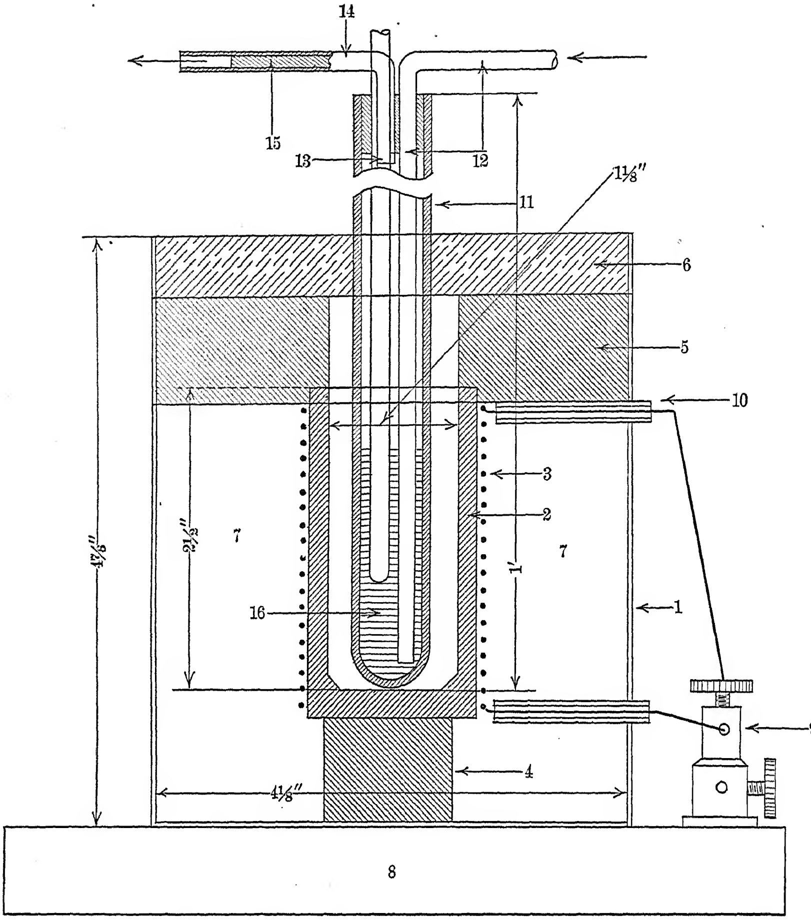

FIG. 4.—SMALLER ELECTRIC FURNACE.

1. Shell of furnace consisting of quart tin can.
2. Fire clay cylinder with spiral thread cut on outside.
3. Resistance wire, German silver, wound in spiral on 2.
4. Block of asbestos board to support heating cylinder.

5, 6. Asbestos board.

7. Magnesia packing.
8. Asbestos board base.
9. Binding post for electrical connection.
10. Clay pipe-stem insulator for resistance wire.
11. Silica tube sealed at lower end.
12. Silica tube for delivering gases to bottom of 11.
13. Pyrometer tube of silica.
14. Glass tube for delivering gases from 11 to absorption tube.
15. Plug of glass wool in 14 to prevent salt from being carried mechanically into absorption tube.
16. Position of sodium chloride in silica tube 11.

The effect of a more intimate contact between the sulphur trioxide-air mixture and the salt was studied by constructing a small furnace and substituting it in the place of the larger tube furnace in the train of apparatus shown in Fig. 2. In the small furnace the salt was contained in the closed end of a silica tube, and through the column of salt passed the silica pryometer tube and a silica tube for the delivery of the gas mixture. The details of the arrangement are shown in Fig. 4. It is apparent that in this furnace the gas mixture was brought into much more thorough contact with the salt than in the one previously used. With 5 g. of finely pulverized sodium chloride in the tube the sulphur trioxide and air mixture was passed for 15 min., while the furnace temperature was held at 593° C. During this time 0.1248 g. of silver chloride was precipitated in the absorption tube, corresponding to 0.0308 g. of chlorine. A comparison of the amount of chlorine obtained in this experiment with the amount obtained from the tube furnace in the same time and at practically the same temperature (see the preceding table) shows, as might be expected, that better contact between the reacting substances greatly increases the rate of chlorine evolution.

Since the rate of chlorine production in the small furnace was considerably greater than in the tube furnace, further tests were made to determine the minimum temperature at which chlorine will be produced by the action of sulphur trioxide and air upon sodium chloride. About 5 g. of the pulverized salt was placed in the furnace tube, and, with the gas mixture passing, the temperature of the furnace was gradually raised. The indicator became turbid at a furnace temperature of 505° C. The furnace was then allowed to cool, fresh indicator was provided, and the heating was repeated. In three successive tests of this kind the first turbidity was noticed in the indicator when the furnace temperature was between 500° and 505°. It is therefore concluded that chlorine is first evolved at a sufficient rate to be detected by the methods employed at a temperature of 500° C. It seems quite possible that by mixing the salt with other materials, as it is mixed in the roasting furnace, catalytic effects might be produced, which would increase the velocity over that which was observed with the pure material. This point was not studied.

Reaction of Sodium Chloride, Silica, and Oxygen.—The second reaction studied was that involving the combination of sodium chloride, silica, and oxygen to form sodium silicate and chlorine according to the equation, $4\ NaCl + 2\ SiO_2 + O_2 = 2\ Na_2SiO_3 + 2\ Cl_2$. It has been supposed that this reaction takes place in the chloridizing roast, but experiments performed as described below failed to show any such reaction even up to the temperature at which the salt melts. The tests were made with a mixture of pure sodium chloride and pure silica in the proportion of 1.71 g. of the former to 1 g. of the latter. These

proportions would yield a silicate of the formula Na_2SiO_3. The thoroughly ground mixture was placed in the silica tube of the small furnace, and the pyrometer tube and the tube for the delivery of air were arranged as shown in Fig. 4. Starting with the furnace cold and heating it slowly, air from the drying train was drawn through the salt and silica mixture and then through the absorption bulb. Up to the temperature at which the salt melted, 800° C., the indicator remained perfectly clear. Thinking that the presence of water vapor would aid in the decomposition of the salt with the formation of hydrochloric acid, a second similar test was made in which the air was passed through a wash bottle containing distilled water before it was admitted to the furnace. The result was the same as before; up to the temperature at which the salt melted no chlorine or hydrochloric acid was detected in the indicator. Therefore it is concluded that at temperatures below its melting point salt is not decomposed to an appreciable degree by the combined action of silica and atmospheric oxygen, either in the absence of water vapor or in its presence.

The Chloridation of Copper Oxide and Nickel Oxide

The study of the chloridation of the oxides of nickel and copper was undertaken in order to determine the conditions most favorable for the chloridation of each, and thus to demonstrate whether or not a separation of these metals can be effected by the use of the chloridizing roast. In particular would such a process be useful in treating heavy pyritic ores containing copper and nickel. Such ores occur at Sudbury, Ontario, and are there treated by heap roasting followed by partial pyritic smelting for copper-nickel matte. The matte thus produced is concentrated in a basic converter to a matte containing 80 per cent. of combined nickel and copper with less than 1 per cent. of iron. The converter slag is resmelted in reverberatory furnaces. By this method of treatment all of the iron of the ore enters the waste slags of the furnaces and is therefore lost. If the copper can be removed from such ores and the iron and nickel left in a residue of such character as to be suited to blast-furnace smelting for nickel-bearing iron, the iron would be saved and the value of the ore would be correspondingly increased. In addition to saving the iron, there is of course the possibility of utilizing the sulphur for the manufacture of sulphuric acid, where such utilization is warranted by local conditions and markets.

The chemical reactions involved in the chloridation of copper in burnt pyrite have been investigated by E. Kothny,[2] who concludes that reactions represented by the following series of equations are largely responsible for the chloridation of the copper:

[2] *Berg und Hüttenmännisches Jahrbuch*, vol. lviii, p. 97 (1910). *Metallurgie*, vol. viii, No. 13, p. 389 (July 8, 1911).

$$2 FeS_2 + 7 O_2 = Fe_2(SO_4)_3 + SO_2.$$
$$Fe_2(SO_4)_3 + 6 NaCl = 3 Na_2SO_4 + 2 FeCl_3.$$
$$3 CuO + 2 FeCl_3 = 3 CuCl_2 + Fe_2O_3.$$

Further points shown by Kothny are, that the reaction $4 FeCl_3 + 3 O_2 = 2 Fe_2O_3 + 6 Cl_2$ proceeds rapidly from left to right when a current of dry air is passed over the heated ferric chloride, while the similar reaction between cupric chloride and oxygen, $2 CuCl_2 + O_2 = 2 CuO + 2 Cl_2$, is comparatively slow under the same conditions; also that the dissociation of cupric chloride into cuprous chloride and chlorine, $2 CuCl_2 = Cu_2Cl_2 + Cl_2$, does not occur at temperatures of 340° to 550° C. when the cupric chloride is contained in a mixture in which an excess of sodium chloride is present. It is important that the two latter reactions be prevented from progressing in the left to right direction, because by so doing they tend to defeat the object of the roast. Since both are doubtless reversible, the presence of chlorine in the furnace gas and in the interstices of the roasting ore will tend to send these reactions in the right to left direction. Therefore it seems advisable to maintain as high a concentration of chlorine in the furnace gas as is possible without an undue consumption of salt.

In the experiments performed by the writer mixtures of copper oxide and nickel oxide with ferric oxide, silica, ferric sulphate, and sodium chloride were subjected to heatings at various temperatures and for different times in an electric tube furnace, a current of dry air being passed over the mixture during the heating. The compositions of the mixtures are shown in the following table:

Constituents	Mixture No. 1		Mixture No. 2		Mixture No. 3	
	Grams	Per Cent.	Grams	Per Cent.	Grams	Per Cent.
Cupric oxide............	0.2503	5.00 Cu	....	....	0.2503	5.00 Cu
Nickel oxide............	....	....	0.2551	5.00 Ni	0.2551	5.00 Ni
Ferric oxide............	1.778	44.5	1.639	41.0	1.523	38.1
Silica..................	0.400	10.0	0.400	10.0	0.400	10.0
Ferric sulphate.........	0.838	21.0	0.910	22.8	0.838	21.0
Sodium chloride.........	0.734	18.3	0.796	19.9	0.734	18.4
Total weights..........	4.0003		4.0001		4.0004	

In composition these mixtures are intended to represent dead-roasted sulphide ores containing copper or nickel or both, to which have been added ferric sulphate and sodium chloride. The percentages

of copper and nickel in the mixtures are somewhat higher than in the ores that these mixtures are intended to represent. However, the quantity of mixture that could be used in an experiment was limited by the size of the boat and of the furnace to 4 g. Since it was not considered advisable to work with less than 0.2 g. of copper or nickel, the mixtures were made to contain 5 per cent., as is shown in the table. In order that no uncertainty should arise in regard to the amount of copper or nickel present in each experiment, the exact amounts of the ingredients of the different mixtures were weighed out and material for each experiment was mixed separately in an agate mortar. In practical applications of the chloridizing roast it is customary to produce the ferric sulphate, necessary for the decomposition of the salt, in the furnace by the oxidation of pyrite. In laboratory work with the small amounts of material used in these experiments it is more convenient to add prepared ferric sulphate, and by its use to avoid any irregularities that may arise from the roasting of the pyrite.

Mixture No. 1 contains double the amounts of ferric sulphate and sodium chloride necessary to convert all of the copper into cupric chloride. Mixture No. 2 contains double the amounts of these ingredients necessary to convert all of the nickel into nickel chloride. Mixture No. 3 contains enough ferric sulphate and sodium chloride to chloridize both the nickel and the copper.

Preparation of the Materials.—Of the materials used in the mixtures the ferric oxide, silica, and sodium chloride required no special preparation. The ferric oxide was that prepared by Kahlbaum, and showed no traces of copper or nickel when tested qualitatively. The silica was prepared by grinding clear white quartz in a porcelain jar mill. The sodium chloride was Merck's C. P., and was prepared by heating to about 600° C. and then pulverizing in an agate mortar.

The nickel and copper oxides were prepared by calcining their respective nitrates. Both nitrates were dehydrated over a gas flame and then heated in a muffle until decomposition was complete. The oxides thus produced were boiled with water to remove soluble material, and, after filtering and washing, the oxides were again heated in the muffle. The averages of three electrolytic determinations on each of the materials are given below and compared with the theoretical composition of the pure compounds.

	Per Cent.
Copper oxide prepared	79.88 Cu
CuO pure	79.89 Cu
Nickel oxide prepared	78.41 Ni
NiO pure	78.58 Ni

Pure anhydrous ferric sulphate was prepared from the C. P. salt of Baker & Adamson. The raw material contained, as undesirable impurities, sulphuric acid, nitric acid, and water. A part of the vol-

atile impurities was removed by heating over a gas flame. Then in order to free the salt from acid and water it was heated to 500° C. for a few minutes. The heating was done in a closed tube of hard glass, the closed end of the tube containing the salt being heated in the small furnace shown in Fig. 4, while the upper and open end of the tube was heated with a gas flame to prevent the condensation of the impurities. About 20 g. of the anhydrous ferric sulphate was prepared in this manner; it was in the form of a fine cream-colored powder. The analysis is shown

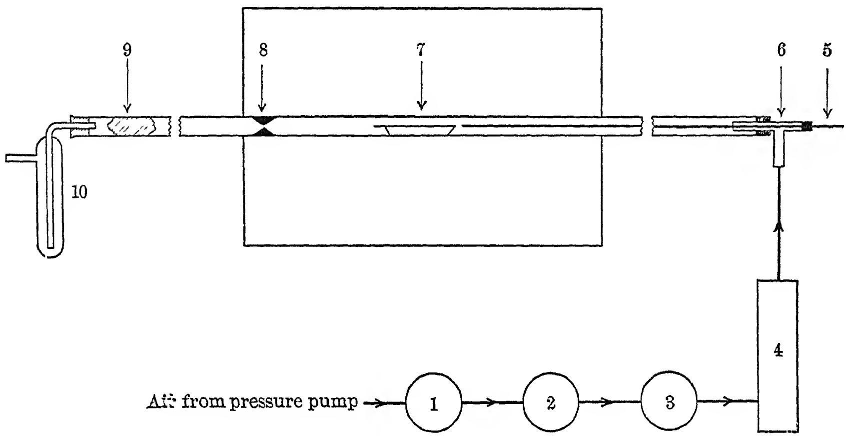

Fig. 5.—Diagram of Apparatus Used in Roasting Mixtures.

1. Wash bottle containing potassium permanganate.
2. Wash bottle containing potassium hydroxide.
3. Wash bottle containing sulphuric acid.
4. Drying tower containing phosphorus pentoxide.
5. Silica tube containing pyrometer wires.
6. Glass T through which pyrometer tube and air pass into the furnace tube.
7. Porcelain boat containing mixture to be roasted.
8. Point in the tube where most of the volatilized material condensed.
9. Plug of glass wool to provide condensation surface for volatilized material.
10. Railroad tube containing water.

The furnace and porcelain tube are the same as those shown in Fig. 1.

below compared with the theoretical composition of the pure compound; the average of two analyses is given.

Prepared ferric sulphate: $Fe_2O_3 = 40.32$, $SO_3 = 59.68$ per cent.
$Fe_2(SO_4)_3$ pure: $Fe_2O_3 = 39.93$, $SO_3 = 60.07$ per cent.

From the comparison it is evident that only an inconsiderable amount of the sulphate was decomposed during the process of drying.

Method of Conducting the Experiments.—The furnace shown in Fig. 1 was connected with other apparatus as shown in Fig. 5. The furnace was heated to the desired temperature with the current of dry air passing through it, then the boat with its contents was placed in the position shown. A plug of glass wool was inserted into the end of

the tube for the purpose of providing a large surface upon which volatilized copper chloride might condense. The gases from the tube were passed through a railroad tube containing water in order to catch any copper salt that did not condense in the tube or on the glass wool.

During the first 10 min. of heating, chlorine and sulphur trioxide were evolved freely. After this length of time the rate of evolution of these gases usually diminished perceptibly, and continued to diminish until at the end of 4 hr. the air coming from the tube contained only small amounts of them. The time of heating was in most cases 4 hr.; in a few it was only 2 hr.

The products from an experiment performed as described were the volatilized material that had been carried by the air current to the cooler part of the furnace tube, where it condensed, and the residue remaining in the boat. The latter was in the form of a dark-brown porous cake, which could be removed from the boat in a single piece, but which was soft and could be easily pulverized. Most of the material volatilized from the boat condensed in a somewhat hard ring at the point (8) shown in Fig. 5; a small amount of it condensed on the glass-wool plug; in no case was a trace of copper or nickel found in the railroad tube. The removal of the ring of condensed material from the wall of the tube was at times quite difficult, and particularly so during the last series of experiments with mixture No. 3. In all cases it was necessary to place a rubber stopper in one end of the tube, then to introduce about 25 cc. of glass beads together with the solvent, and then, after inserting a stopper in the other end, to shake the tube and contents vigorously. The beads aided in removing the deposit from the wall of the tube and thus made the action of the solvent more rapid. One or two such treatments with ammonia followed by one with nitric acid were generally used. Even this treatment failed to remove all of the material after the glaze on the inner surface of the tube had been partly destroyed by the combined actions of the deposit and of the beads, so that in the later experiments all of the copper in the volatilized product was not recovered. An interesting point in regard to the volatilized material is that in no case was any appreciable amount of iron found in it. This fact indicates that if the volatile ferric chloride is formed during the roast, it is decomposed before it can pass from the roasting material into the gases.

The products from each experiment were treated as follows: In the volatilized material copper or nickel or both were determined electrolytically. The material in the boat was boiled with water, filtered, and washed, in order to remove water-soluble copper or nickel. Since this treatment would not remove cuprous chloride if present, the residue was further treated by boiling with saturated sodium chloride solution, after which it was again filtered and washed. The brine solution in all cases contained only small amounts of copper, and accordingly was

not analyzed separately, but was added to the solution obtained from the water treatment. The copper in the combined solutions was determined electrolytically, after evaporating with sulphuric acid to white fumes, adding 3 cc. of nitric acid, and diluting to about 200 cc. with water. When nickel was present it was also determined electrolytically. In the case of the experiments with mixture No. 2, which contained no copper, the brine treatment was omitted. The residue from these operations, together with the incinerated filters that had been used, was boiled with three successive portions of concentrated nitric acid. This treatment served to remove the copper oxide without dissolving enough of the iron oxide to interfere with the subsequent electrolytic determination of the copper. No attempt was made to determine nickel in the residues. In treating the residues with water and with brine it was observed that not more than traces of iron were dissolved in any case.

Results of the Experiments.—The results of a number of experiments with the three mixtures are given in the following table:

Mixture No.		Temperature, Degrees C.	Time, Hours	Per Cent. Volatilized		Per Cent. Soluble in H_2O and Brine		Per Cent. in Residue, Cu	Per Cent. Accounted for, Cu
				Cu	Ni	Cu	Ni		
1	Cu only..	500	4	24.9	......	36.2	......	29.0	90.1
1		600	2	29.9	......	29.5	......	33.1	92.5
1		600	4	68.1	......	7.5	......	17.2	92.8
1		700	2	71.6	......	15.4	......	9.0	96.0
1		700	4	92.5	......	0.9	......	1.9	95.3
2	Ni only..	500	4	......	trace[a]	......	6.2	....	....
2		550	4	......	trace	......	17.7	....	....
2		600	4	......	trace	......	13.0	....	....
2		650	4	......	trace	......	1.5	....	....
2		700	4	......	trace	......	0.8	....	....
3	Cu and Ni	600	4	47.7	trace	21.5	2.3	12.9	82.1
3		650	4	71.4	2.9	2.7	0.4	6.5	80.6
3		700	4	lost	0.8	4.5	1.0	4.1	....
3		700	4	67.6	1.6	8.5	1.2	3.4	79.5
3		700	4	68.1	2.7	1.3	trace	5.1	74.5

[a] The amounts of nickel volatilized in the experiments with mixture No. 2 were too small to be estimated quantitatively. The volatilized portion was accordingly added to the solution containing the water-soluble nickel, and the combined solutions were analyzed for nickel. In the experiments with mixture No. 2 the brine treatment was omitted.

In the first series of experiments the amounts of copper accounted for vary from 90.1 to 96.0 per cent., while in the third series still less of the copper was accounted for. This is due to the increasing difficulty that was encountered in removing without loss the volatile products from the tube; this point has been mentioned in a preceding paragraph.

In reviewing the results of the experiments it is necessary to bear in mind the facts that only small amounts of material were used, that the time of treatment in each case was comparatively short, and that the results cannot be directly compared with those secured when a chloridizing roast is performed on a large scale because of the differences in the conditions. It was the intention to study the behavior of copper oxide and nickel oxide when mixed with iron oxide and subjected to heating under the conditions that favor the chloridation of the metals. It is necessary to consider as chloridized that part of the copper and nickel which was volatilized, because neither the sulphates nor the oxides of these metals are volatile under the conditions existing in the experiments.

The experiments with mixture No. 1 show that the completeness of chloridation of the copper increases with increasing temperature and with increasing time. An interesting point is the fact that in the last experiment with this mixture 92.5 per cent. of the copper was recovered from the volatile product. Experiments with mixture No. 2 containing nickel oxide show the behavior of the nickel under conditions similar to those to which the copper oxide was subjected. Only traces of nickel were found in the volatile products, as is stated in the footnote to the table. The maximum production of water-soluble nickel resulted at 550° C., the nickel thus extracted amounting to 17.7 per cent. The experiments at higher temperatures showed decreasing amounts of water-soluble nickel. Roasting for 4 hr. at 700° C. produced only 0.8 per cent. of water-soluble nickel. Whether or not the nickel extracted in these experiments was in the form of chloride or sulphate was not determined; the conditions were favorable for the production of either compound.

Considering the results of these two series of experiments, it appears that nickel is not rendered water soluble nor is it volatilized to any considerable extent when the temperature of the roast is above 650° C., while such temperatures are quite favorable to the chloridation of copper. In confirmation of this conclusion we have the results of the third series of experiments, which show a maximum of 3.3 per cent. of the nickel converted into water-soluble and volatile compounds at 650° C. The increased tendency toward volatilization exhibited by the nickel in the presence of copper is probably due to the more or less mechanical influence exerted on the nickel chloride by the volatilization of the copper chloride.

These laboratory experiments of course do not prove that roasted pyritic ores containing copper and nickel can be treated commercially

by chloridizing the copper and leaching it from the iron and nickel oxides, leaving a product suitable to be smelted to nickel-bearing pig iron. That could only be demonstrated by carefully conducted experiments on large quantities of the ore in question. However, the experiments do prove the possibility of effecting such a separation and indicate the temperatures at which the roast should be conducted in order to secure the desired results.

Discussion

H. O. Hofman, Boston, Mass.—At the close of the paper Mr. Dudley said that his experiments in the laboratory had shown that it was possible to separate nickel from copper by means of a chloridizing roasting, but that in order to demonstrate a process of this kind, work upon a large scale would have to be carried out.

I think it was in 1913 that experiments of this kind were carried out at the works of the Pennsylvania Salt Manufacturing Co., Philadelphia, in a Wedge furnace, and at Sault Ste. Marie in a Sjöstedt furnace.

They showed a satisfactory separation of nickel and copper. The ore was first subjected to an oxidizing roast, then mixed with pyrite and salt, and given a chloridizing roast.

A. H. Carpenter published the details of this work in the *Engineering and Mining Journal*, vol. xcvii, p. 1085 (1914). The conclusion arrived at is, that if you subject sulphide ore to an oxidizing roast, finishing at the temperature at which all nickel sulphate is dissociated, *i.e.*, 764° C., then mix it with pyrite and salt, and subject the mixture to a chloridizing roast between 600° and 700° C., you can chloridize a high percentage of the copper without chloridizing any of the nickel, and obtain thereby a satisfactory means of separating the two.

These experiments have also shown that if in the oxidizing roast the temperature is not carried sufficiently high, so that water-soluble nickel sulphate remains in the ore, this nickel sulphate will be chloridized.

Copper Smelting in Japan*

BY MANUEL EISSLER, PARIS, FRANCE

EDITED BY BURR A. ROBINSON, ASSISTANT SECRETARY

(New York Meeting, February, 1915)

Introduction

The history of copper metallurgy in Japan goes back into remote ages; of this there is abundant proof, and that the working of this metal is closely connected with the artistic development of this nation is evidenced by some of the glorious monuments which have been produced by these ancient metal workers. The traveler sees to-day in a perfect state of preservation bronze statues 150 ft. high, and over, 300 ft. in width, and 170 ft. in depth, delicately chased and beaütifully marked, and when one reflects that some of them date back to the eighth century A. D., the question naturally suggests itself, how did these ancient metallurgists with their crude and primitive appliances accomplish this stupendous task?

The bodies of these images are formed of bronze castings 10 to 12 in. thick, which were gradually built up and soldered together, but other pieces of magnitude, like the heads and other portions of the body are single castings. In many cases another method of construction must have been adopted. The walls of the molds were gradually built up and as the lower portion of the castings cooled, the walls were raised higher and higher and fresh metal must have been added, instead of constructing the whole mold first and then making the casting in one piece.

There are visible on these statues, still traces of substantial gilding, and how the pious Buddhist priests who were the constructors of them got the precious metal, is preserved in legendary tales.

As a matter of interest I will mention that mines in Japan are not generally worked as joint stock enterprises, but are mostly family properties which have descended to the present owners from their ancestors. Only in recent years have some new discoveries, in rare

* Based on notes taken during a journey to the principal mining districts of the Japanese Empire in 1906 and 1907.

cases, sought the assistance of foreign capital. Only one mining company has its shares dealt in on the stock exchange and mines are worked like other industrial undertakings. Owing to the abundance of cheap labor they are yielding profits on exceedingly low-grade ores. The present production of copper in Japan is about 45,000 tons a year and from what I have seen of the country I believe it will gradually increase.

What actuated me in presenting this modest work to our Institute was my desire to place before the members a description of how old methods are still adhered to, even at some of the larger mines, where they are installed alongside the most modern and improved American and European methods. It was particularly this commingling of the old with the new which interested me and may also prove of interest to others.

I take occasion to thank all those gentlemen in Japan who have given me the opportunity of visiting their mines and who extended to me their kind hospitality, and I am specially indebted to Dr. Abe and to Dr. Watanabe for their introduction to the mining community of Japan.

This paper shows that Japan understood hundreds of years ago how to extract copper and other metals, and produce with them works of art which will stand as everlasting monuments to prove the high culture of this nation in past ages, when some of our European countries were still in darkness, and when Grecian and Roman art was forgotten.

I. Smelting at the Besshi Works of the Sumitomo Company

The mineral at the Besshi mine is an intimate mixture of iron and copper pyrites and contains considerable magnetic iron and silica. The average contents of the ores are about 4 per cent. of copper. The quantity of ore smelted in 1905 amounted to 32,000,000 kwans (1 kwan = 8.26 lb.) or 132,160 tons and the production of copper amounted to 5,200 tons, showing an extraction of 3.93 per cent.; consequently 25.4 tons of ore were required to produce one ton of copper. On the basis of 340 working days, 388 tons are smelted daily. There are no precious metals in the Besshi ore.

The smelting works are situated on Shisaka Island in the Inland sea and distant 10 miles from the mines which are on Shikoku mainland. The ores are brought in boats of 30 tons capacity, and generally six of these are propelled by means of a tug boat from the landing stage on the island. The ore is hoisted in 1-ton trucks on a double-track line up an incline by means of a winding engine to a station 158 ft. above sea level and is distributed from here to the roasting stalls which are disposed along the slope of the hill. There are 635 ore-roasting stalls and 126 matte-roasting stalls.

The roasting stalls, which are made of slag bricks, are 13 ft. long, 5.5 ft.

wide and 6 ft. deep. The front walls are 3 ft. thick and the division walls 2.5 ft. In the front there are two charge openings 4 ft. high and 3 ft. wide. There are nine small openings in the back walls which communicate with the flue. The four benches or terraces on which the stalls are built have each a length of 448 ft.; the width of the first bench is 88 ft. The flues into which the stalls debouch are 5 ft. 8 in. wide and 6 ft. in height. The transverse flue, into which the longitudinal flues debouch, is 6 ft. wide and 9 ft. 9 in. high and connects with a dust chamber 10 ft. wide and 21 ft. high leading to the smoke-stack.

For roasting, wood and 5 per cent. of the weight of ore in coke are employed. The ore-roasting stalls hold 35 tons each and the roast is finished in 45 days. The matte-roasting stalls hold 30 tons each; roasting is finished in two weeks. For taking the roasted matte and ores down hill to the smelters, incline tracks working automatically are employed. The stalls for matte roasting are under sheds, those for ores in the open.

The Blast Furnaces

There are three blast furnaces in the ore-smelting house, which have each a smelting capacity of 200 tons of ore, and as only two furnaces are at work and one is held in reserve, the quantity of ore smelted daily is about 380 to 400 tons. Owing to the basic nature of ore, 60 tons of siliceous rock are added; the daily consumption of coke is about 70 tons or 15 per cent. of the furnace charge. The siliceous rock is mined from an adjacent island and is barren. The ore after stall roasting contains from 10 to 12 per cent. of sulphur.

The three water-jacket furnaces are each 16 ft. long and 3 ft. 3 in. wide, equipped with ten 4-in. tuyères on one side, 12 tuyères on the other or back portion and one tuyère on one of the ends. The front consists of six water jackets, 5 ft. 8 in. high, 3 ft. wide, with a water space of 4 in. These water jackets are built at the workshops of the company in the Niihama mining village. The tuyères are all provided with cutoff valves. The water jackets are held together by a ring made of steel rails. From the center of the tuyères to the feed floor is only 7.5 ft. The cast-iron drop bottom of each furnace is held in position by jack screws. The top portion of the furnace is supported by 10 iron columns. The air pipes around the furnaces are 12 in. in diameter. The cold-water pipes to jackets are 2.5 in. in diameter and so are the outlet pipes on top of jackets. The slag spout is water jacketed with coils of pipes. Each water jacket is provided with two hand holes and two valves for drawing off the sediment. As sea water is used in the jackets, great attention is paid to the cleaning of jackets. When cleaning out the jackets, which is done once a month, a hose is attached to one of the valves and water under pressure is pumped into them to stir up the sediment. The water

used in the water jackets is pumped from the sea into a large brick reservoir holding 253,600 gal.

On top of each furnace are three short iron chimneys, 3 ft. in diameter and lined with brick, leading into a circular dust chamber 11 ft. in diameter lined with fire clay which runs the total length of the three furnaces and on top of them. This dust chamber is cleaned out with compressed air.

The slag and matte run continuously into a circular forehearth, 10 ft. in diameter and 4 ft. high, which is water jacketed. In the bottom are several layers of fire bricks. The matte can be tapped from both sides of the hearth. There are also several hand holes near the bottom of this forehearth to clean out the sediment. Matte is drawn every 4 hr. Any slag collected from the matte pots is resmelted.

The daily production is about 60 tons of matte, or 15 per cent. of the weight of ore smelted and the same contains from 27 to 30 per cent. of copper. Each furnace requires 4,000 gal. of jacket water per hour and experiments are now being made with water jackets made of copper.

The matte composition is as follows: Iron, 40.22 per cent.; sulphur, 18.88 per cent.; copper, 30.78 per cent. The slag composition is: SiO_2, 35.48 per cent.; FeO, 51.73 per cent.; Al_2O_3, 7.35 per cent.; copper, 0.429 per cent.

Smelting for Second Matte

The first matte produced is broken up by hand labor and goes to the matte-roasting stalls and after roasting still contains 12 to 13 per cent. of sulphur. It is then smelted in blast furnaces, situated in a separate building. This concentrated or second matte contains 70 per cent. of copper. The molten matte from these furnaces is run into reverberatory furnaces, which convert it into blister copper 98 to 99 per cent. fine. The blister copper is taken to another building and smelted in reverberatory refining furnaces and made into fine rose copper of 99.7 per cent. fineness.

There are three circular matte-smelting furnaces, which have a capacity of 30 tons each; two are constantly in operation, while one is held in reserve or is undergoing repairs. With the matte is also smelted all refuse from the blister and refining furnaces as well as all skimmings and dross. To the charges is added 15 to 16 per cent. of the siliceous rock as above mentioned and 17 per cent. of coke is used. Why the owner of the mine does not buy siliceous copper ores to utilize in place of waste rock, I do not know, unless there is a lack of such material in the neighboring mines. These matting furnaces are 3 ft. in diameter and have eight 2.5-in. tuyères each. The water jackets are 4 ft. high and the distance from center of the tuyères to bottom of the jackets is 6 in. The crucible shell is 4 ft. deep. The distance from the tuyère level to feed floor is 7.5 ft.

The slag is run into pots and after cooling the bottom cone is broken off and returned to the ore-smelting furnaces. The same is also done with the slag which is drawn off from the forehearth in the first matte-smelting department just before tapping the matte. The slags from the No. 2 matte smelting, which are thus returned to the ore-smelting furnaces, contain 1.5 per cent. copper.

The crucibles of the matting furnaces are lined with fire bricks and with a layer of brasque composed of charcoal powder and clay. These furnaces are erected on an elevated platform, so that there is sufficient grade for the molten material to run direct into the blister furnaces. In the dust chamber 10 per cent. of the weight of material is collected and the dust contains 30 per cent. of copper; it is melted down and returned to the furnace.

The Blister Furnaces

There are six reverberatories, two for each cupola. Their dimensions are 16 ft. 8 in. by 7 ft. 8 in. with fireplaces 5 ft. 5 in. by 3 ft. 5 in. The matte runs alternately into these furnaces by means of sheet-iron gutters lined with clay and charcoal, which are suspended on chains. Compressed air at 6-lb. pressure is blown in on both sides of the reverberatory furnaces. Each furnace holds 3 tons of matte and every 12 hr. 1.8 tons of blister copper is drawn. The furnaces are filled with the molten material from an opening in the top. The quantity of fuel required is equal to 10 per cent. of the weight of the matte. The slags from these furnaces go back to the matting furnaces. The chimneys from the six furnaces debouch into an overhead flue, or dust chamber, supported on independent iron girders. The top portions of the chimneys, where they connect with the dust chamber, are made of boiler iron. The blister copper is ladled out, cast into ingots and sent to the refining furnace. The ingots weigh 40 lb., and the ratio of blister copper to refined copper is 95 per cent.

Refining Furnaces

There are two refining furnaces having each a capacity of 15 tons in 24 hr. The consumption of fuel as given to me was: Coal 32 per cent.; charcoal, 1.6 per cent.; poling wood, 6 per cent.; total, 39.6 per cent. of the weight of copper, which seems to me extremely high.

The sand or ganister for making the bottom of these furnaces comes from the Province of Iwagi in the northern portion of Japan.

For home consumption the copper is cast into square plates 12 by 12 in., weighing 42 *kin* or 54.6 lb. (1 kin = 1.3 lb.). The ingots for export weigh 23 lb. The molds are of copper and rest on a circular table which rotates by mechanical means. A thick muddy water is poured into the hot molds and as the water evaporates a clay lining is left behind which

prevents the ingots sticking to the molds. When ladling the metal into them, a woman drops into each mold a small strip of refined copper, which superstition says imparts great virtues to the metal. This practice dates from mediæval times and is religiously kept up. The Sumitomo brands enjoy a high reputation not only in Japan, but also in other countries.

Dust Chambers, Flues and Chimneys

In the ore-smelting department, the iron flue on top of the blast furnaces is 13.7 ft. in diameter, lined with fire clay, giving an inner diameter of 11 ft.; the length of this iron flue is 153 ft. This flue debouches into a dust chamber 121 ft. in length, 14 ft. wide and 21 ft. high, which connects with another dust chamber 220 ft. long, 11 ft. high and 14 ft. wide. The latter joins the chimney which has an internal diameter of 15 ft. at the base and a height of 210 ft. Its outer diameter is 28 ft. at the base, consequently the brick work is 6.5 ft. thick. At the top of the chimney, the outer diameter is 17 ft. The foundations of the chimney are 35 ft. in diameter. The walls of the dust chambers have a thickness of 1.87 ft. and the arches 1.49 ft.

The flue from the matte-smelting furnaces is also of sheet iron. It connects with a vertical brick flue leading to a dust chamber which is 45.5 ft. long, 9 ft. wide, 15 ft. high and connects with another chamber 76.5 ft. long, two stories high, each story 12 ft. wide and 24 ft. high. A flue 51 ft. long, 7 ft. wide, 10.5 ft. high connects the last chamber with the chimney. The flues from the roasting stalls are also connected with extensive dust chambers leading to an independent chimney 204 ft. high of similar dimensions as described above. The works present an imposing appearance and were erected regardless of expense.

Sea water is used in the boilers and they are cleaned once a month. The mine is provided with extensive machine and repair shops and an iron foundry.

Cement Copper from the Mine Water

The mine discharges about 25 cu. ft. of water per minute, which is passed through a series of brick tanks, lined with cement. There are 152 of these tanks arranged in four rows of 38 each. The dimensions of these tanks are 12 ft. in length, 8 ft. in width and 4.5 ft. in depth.

At the head of the tanks is a large distributing reservoir for settling the sediment. Between the second and third row is a main gutter into which the water is discharged when cleaning up. There is an outer gutter for the first row and one for the fourth. The tanks contain coke and scrap iron in twelve layers arranged alternately. The coke when saturated is sent direct to the smelter, as the cement copper settles in its pores. The yearly production is 225 tons cement copper, which contains 60 per cent. copper.

Remarks and Observations

I have obtained no data as to cost of production. Judging from the character of the ore I think that eventually pyritic smelting will be adopted at these works, but it will be necessary to mix the ores with a siliceous copper ore so as to slag excess of iron and this will materially reduce the cost of making copper. Considering the successful operation of all the other metallurgical operations, I hardly think that, on a small output of say 15 tons copper per day, the discard of the rest of the plant and the introduction of converters would be justified.

II. Smelting at the Ikuno Mines

The metallurgical plant at this mine consists of: Concentration plant; Kiss process plant; smelting works. It is only the last that is of interest to us for the present; the following materials are sent to the blast furnaces: Gold and silver ores from the Tasei mine; copper and silver ores from the Kanagase mine; concentrates and the fines from the copper ore, made into briquettes.

The product is black copper assaying: copper 96 per cent.; silver, 0.8 per cent. or 266 oz. per ton; gold, 0.015 per cent. or 5 oz. per ton.

The various ores which are smelted here come from several mines which are within a radius of 10 miles.

The Tasei mine furnishes a brownish colored quartz, which contains sulphide of silver in association with native gold, a small quantity of pyrite, copper sulphides and blende.

The Kanagase mine furnishes quartzose ores with copper sulphides, some galena, blende and pyrite.

The Otsubushi ores contain metallic silver, galena, blende, argentic sulphide, bornite, some antimony, nickel and cobalt. The galena ores do not seem to carry much silver and good samples of tetrahedrite and argentite are frequent in the mine.

The Mikobata mine carries silver ore with gold.

The Nakase mine furnishes silver ores and gold in association with cupriferous pyrites. Native silver is often found with pyrargyrite.

The smelting as carried out at present is pyritic and the fuel is only added in the outer ring of the furnace, when the same does not run regularly.

The ores contain about 17 per cent. of sulphur, of which 9 per cent. burns and acts as a fuel and 8 per cent. is absorbed in the matte. This system was introduced in 1902 or 1903; prior to that period the ore only contained 9 to 10 per cent. of sulphur. As an average of 17 per cent. of sulphur is not sufficient, pyritic ores are brought from outside mines whenever obtainable and cheap enough to leave a profit.

There are 35 roasting stalls, dimensions 5 by 9 ft., each holding 7 tons. The fuel is mostly pine wood and is used at the rate of 1 per cent. of the weight of ore. The roasting takes one week, but why there is such a large quantity of ore roasted when there is a lack of sulphur, I did not learn.

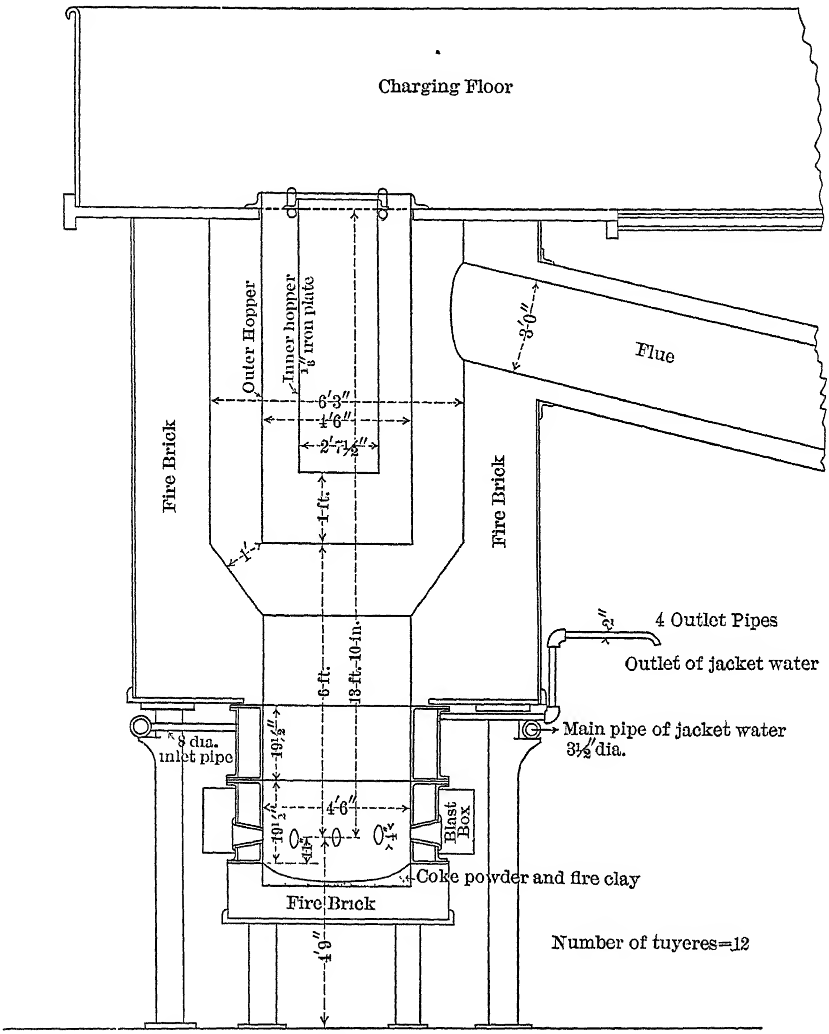

FIG. 1.—SEMI-PYRITIC BLAST FURNACE AT IKUNO WORKS.

The concentrates are made into briquettes by being mixed with 5 per cent. of milk of lime. About 30 women are employed at this work; each woman makes 800 briquettes in 9 hr. or 24,000 are made daily. These briquettes are dried in the main flue leading to the chimney which is on top of the hill. They are cylindrical in shape, 4 in. in diameter

and 5 in. high. The main flue near the briquetting shed is enlarged and divided by a longitudinal wall into two parts, and when one section is being employed the draft is turned into the other section. These drying chambers are 30 ft. in length, 9 ft. in height and 8 ft. in width, each holding 30,000 briquettes and the drying is effected in 34 hr.

The ore smelting is done in three cupolas. The No. 1 furnace is 14 ft. high from tuyère to feed floor. The diameter at the tuyères is 52 in. There are eight tuyères. There is an inner circle to this furnace formed of a cast-iron pipe, 30 in. in diameter, which reaches within 6 ft. of the tuyères, consequently this pipe is 8 ft. in length. Into this inner circle are fed coke and ore rich in sulphur, or the fuel portion of the charge.

The furnace is built of two series of water jackets one on top of the other, and discharges into a square forehearth. The spout connecting furnace with forehearth is also water jacketed. The bottom of the furnace is 10 in. below the tuyères. From the first forehearth the slag discharges into another forehearth and then into the slag pots.

The capacity of the No. 1 furnace is 50 tons of smelting mixture. The furnaces are connected with dust chambers built at the back of the works. The percentage of fuel employed is 9 per cent. The matte produced carries about 40 per cent. copper. The cost of smelting is given to me at 6½ yen per ton (equal to 13 shillings) which includes all the roasting and refining charges.

The furnace charges consist of:

	Kilograms
Copper ores No. 3 carrying 15 per cent. of copper	100
Dressed copper ores	150
Copper ores No. 4 carrying 7 per cent. of copper	100
Raw briquettes, only dried	50
Siliceous gold and silver ores	20
Mabuki-Doko slags	150
Limestone	100
Residues from sulphuric acid works (roasted pyrites with some copper)	290
Matte	100
Coke	95
Total	1,155

The matte composition is as follows:

	Per Cent.
Copper	41.20
Iron	22.60
Sulphur	26.40

The slag composition is as follows:

	Per Cent.
Silica	43
Iron	30
Lime	27

The ores carry on an average 6 per cent. of zinc. The iron that is lacking is supplied from the Osaka Chemical works which sells to various mines the residues of its roasted sulphides, containing about 2.5 per cent. of copper, 45 per cent. of iron and 5 per cent. of sulphur; their cost is about 12s. per ton delivered at the mine. The limestone costs 5s., coke 20s., and charcoal 28s. per ton.

The No. 3 furnace has 12 tuyères, and is 54 in. in diameter. Its capacity is 55 tons of smelting mixture. The pressure of the blast is 28 mm.

The charge for the No. 3 furnace consists of:

	Kilograms.
Copper ore, third grade, 15 per cent. of copper	70
Copper ore, fourth grade, 7 per cent. of copper	120
Dressed copper ore	150
Roasted fine ore, in briquettes	100
Gold and silver ores	20
Mabuki-Doko slag	150
Limestone	100
Roasted pyrites from acid works	290
Roasted matte	100
Coke	100
Total	1,200

The smelting of lead ores is carried out in a separate cupola in admixture with auriferous and argentiferous ores, so as to produce a rich lead bullion, matte and slags. The lead-carrying matte is roasted in separate stalls and resmelted with the lead products from the cupellation furnace. The furnace is generally operated one week in the month. The lead bullion produced in the cupola furnace is sent to the cupellation furnace to produce doré silver, which is sent to the Osaka refinery. The matte from the second smelting is sent to the Mabuki-Doko.

The Mabuki-Doko

The *Mabuki-Doko* method is equivalent to Bessemerizing in a Japanese forehearth. The forehearths are shown in plan and section in Fig. 2.

In the construction of the Mabukis an excavation in the ground about 9 ft. square is made, and lined generally with square blocks of a fire-resisting volcanic stone, care being taken to have a drain in the bottom. The cavity is then filled with a mixture of 30 parts of powdered coke, 6 parts of fire clay, 64 parts of burnt clay, and in a moist state is well tamped in. When dry, crucibles are built up of clay and charcoal forming a series of holes 1 ft. 6 in. in depth, with a diameter at bottom of 2 ft. 6 in., and a diameter at top of 2 ft. 11 in. The cover is also built up of clay on a strong wire netting. It requires considerable experience to build up one of these Mabukis, simple as the operation may appear.

After drying, hot slag is poured in and the crust removed after cooling. A 2-in. iron pipe is built in vertically for escape of moisture. In front of the dome-shaped cover is a 5-in. opening to feed matte and fuel. The air blast is introduced on one side only on top of the charge and the tuyère is bent downward. The matte is poured in in a molten state and char-

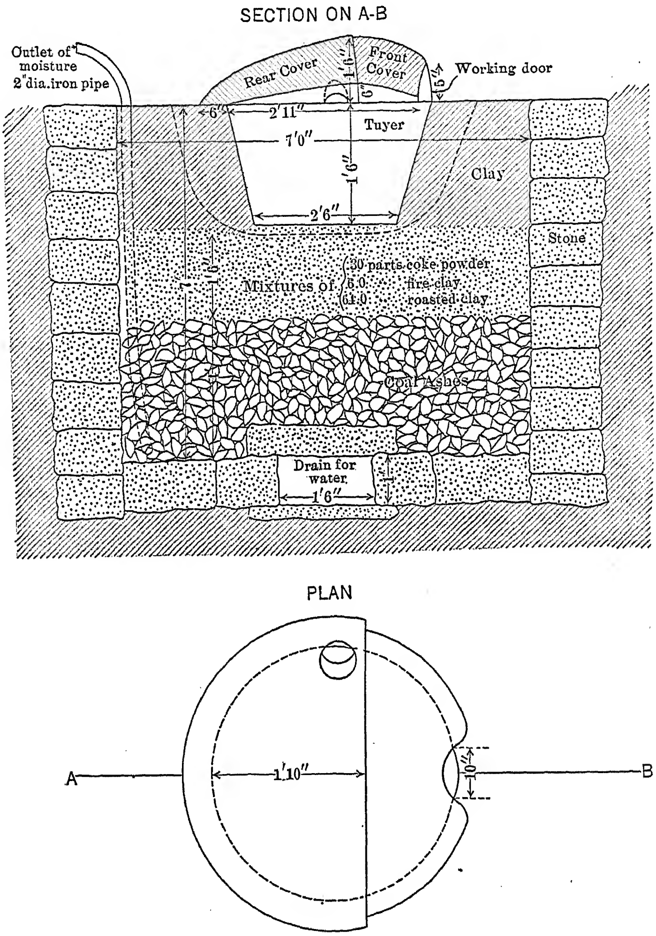

FIG. 2.—SKETCH OF MABUKI-DOKO OR JAPANESE FOREHEARTH.

coal is placed on top; at some mines coke is employed for this purpose. The slag is skimmed every 15 minutes. The idea of introducing the molten matte into the Mabuki originated here, as formerly it was the practice all over Japan to introduce it in lumps and fuse it first.

There are six Mabukis here and the claim is that the loss of copper in this operation is small. The evaporation of the zinc oxide causes silver losses and the claim is that it is recovered in the flues. The copper and

silver diminish in the flue dust, the further the same is collected from the smelting and refining department.

The cost of treatment in the Mabuki is given at 2.5 yen, or 5s. per ton of matte. There are smelted 3 tons of matte daily in each hearth, and generally three of them are in operation dealing with 9 tons. I am told that 6 per cent. of this weight is consumed in charcoal, or only a little over half a ton daily, which seems rather low. As the matte contains 23 per cent. of iron and each forehearth treats 6,000 lb. daily, this would give 1,380 lb. of iron which had partly to be oxidized and partly slagged off by the silica contained in the clay lining. Although the slags produced are very basic, I am mystified at the successful operation of this strictly Japanese method, as the crucibles are not eaten away as fast as it would seem they should.

The cupels of the cupellation furnaces are made of cement. Some iron is introduced during cupellation to absorb sulphur from the press cakes. The slagged material is drawn out through the working doors. The daily capacity of the cupellation furnace is 1.5 tons; one hearth will treat 5 tons before requiring renewing.

The flue dust is collected in a flue 1,560 ft. in length, built on the slope of the mountain. At the point where the flue connects with the chimney, the vertical distance is 600 ft. above the smelting floor level. The dimensions of the flue are 4 ft. wide by 5 ft. high. The chimney is 30 ft. high. In the upper portion, the flue is cleaned twice a year. The lower portion, between the Mabuki-dokos and the briquette drying chambers, is cleaned once a month. The quantity of dust collected in the flue is 0.6 per cent. of the quantity of ore treated. Near the smelting works are dust chambers which are cleaned every two weeks, 5 per cent. of the quantity of ore treated being collected in this section. The flue dust from the Mabuki furnaces contains 4 dwt. of gold, 40 oz. of silver and 24 per cent. of copper; the dust from the chambers near the ore smelting furnaces contains 1 dwt. 8 grains of gold, 5 oz. of silver and 1.5 per cent. of copper. Experiments to ascertain these values and quantities were carried out during two months. The flue connecting the Mabuki-dokos with the main flue is 200 ft. in length.

The production for 1905 was 850 tons of copper, 7,300 oz. of gold, and 200 oz. of silver.

The machinery in the smelting works includes two Baker blowers operated by a 4-ft. Pelton wheel at a water head of 100 ft. and one Root blower.

A 12-in. pipe conveys the air to a reservoir of boiler iron, and from it a 14-in. main takes it to the furnaces to be distributed by 10-in. pipes. The angles of these pipes are of cast iron. There are three Chilean mills for grinding the ash which is used in the Mabukis.

Analysis of Ores Treated at the Ikuno Works

	Au, Oz.	Ag, Oz.	Cu, Per Cent.	Pb, Per Cent.	Zn, Per Cent.	Fe, Per Cent.	S, Per Cent.	CaO, Per Cent.	SiO_2, Per Cent.
Gold and silver ores from Tasei mine	8.8	22.4	0.045	0.01	Trace	4.966	4.739	1.730	85.70
Gold and silver ores from Kasei mine	0.7	88.0	1.870	0.57	......	6.730	3.440	4.140	75.85
Copper ore, 3d grade..............	....	23.0	16.400	1.57	6.21	16.640	25.400	2.910	30.46
Copper ore, 4th grade..............	....	10.0	7.000	1.01	11.59	7.620	11.700	3.890	67.56
Concentrates from copper ore.......	....	11.667	7.900	2.11	14.70	13.500	15.100	1.100	49.23
Raw briquettes....................	0.25	14.0	3.400	2.70	13.927	15.680	16.900	5 397	40.36
Roasted briquettes................	0.3	16.0	3.680	2.81	10.309	17.020	8.200	6.111	42.84
Slag from ore smelting.............	....	0.7	0.412	......	5.770	22.830	......	7.500	42.20
First matte from ore smelting.......	1.7	78.0	41.200	5.50	4.200	22.600	26.400	......	......
Residues from sulphuric acid works	....	......	2.600	......	......	57.420	5.300	......	7.50
Limestone	....	......	......	......	......	......	......	50 900	2.65
Mabuki slag.....................	0.15	10.0	5.700	2.40	2.60	42.10	6.740	......	22.40

III. Smelting at the Ashio Works of the Furikawa Mining Co.

The mines occur in eruptive rocks, such as andesite and trachyte, which have here found a vent through the Archean and Paleozoic rocks, the greater portion granite, which surround on all sides the cupriferous zone embracing the complete system of the Ashio copper veins.

The first-class ores as they come from the mine contain 13 per cent. of copper, and they are enriched by hand picking to 14 per cent. The second-class ores contain on an average 1.5 per cent. and are enriched by concentration to 10 per cent.

There is produced daily on an average 130 tons of first-class ore, which is sorted down to 111 tons; the 450 tons of low-grade ore produce 34 tons of concentrates assaying 10 per cent. of copper. The total sent daily to the smelter is therefore 145 tons. Hence it takes 13.24 tons of low-grade ore to make 1 ton of concentrates and the following calculation shows that the loss in concentration is 50 per cent.:

	Metallic Copper, Tons
450 tons of ore, 1.5 per cent. copper, contain..........	6.75
34 tons of concentrates, 10 per cent. copper, contain .	3.40
Loss in concentration	3.35

The ore going to the smelter averages 13 per cent. of copper and therefore the daily production should be 18.85 tons copper, minus the loss in treatment.

The proportion of mine ore treated daily is:

	Tons
Lump ore, 15.7 per cent. Cu	61.0
Grain ore, 13.3 per cent. Cu	21.0
Fine ore, 11 3 per cent. Cu	61.0
Pulp ore, 6.0 per cent. Cu	2.5
Cement copper, 50.0 per cent. Cu	1.0
Total	146.5

Pulp ore is obtained from settling ponds connected with the concentration plants, where the richer portions of the slimes are collected.

Six Blast Furnaces in Operation

There are six smelting furnaces with dimensions as follows:

No. 1. 17 ft. long by 40 in. wide, with 24 6-in. tuyères.
No. 2. 8 ft. long by 36 in. wide, with 14 4-in. tuyères.
No. 3. 8 ft. long by 36 in. wide, with 14 4-in. tuyères.
No. 4. 8 ft. long by 36 in. wide, with 12 3.5-in. tuyères.
No. 5. 67 in. long by 36 in. wide.
No. 6. 67 in. long by 36 in. wide.

The wind pressure at the furnaces is 35 mm. The feed floors are 8 ft. above the tuyère levels. The main air pipe is 36 in. in diameter. Five Root blowers are driven by a Pelton wheel, with a water pressure of 63 lb. per square inch; they work at a wind pressure of about 40 mm. In winter the blowers are driven electrically. It takes 110 h.p. to run the five blowers. A vertical air compressor is also operated by a Pelton wheel and requires 134 h.p. to furnish air up to the required pressure. The compressed air is required for Bessemerizing. There is a separate Root blower for No. 2 furnace.

The fine concentrates are made into briquettes. These briquettes, while damp, are covered with matte by a patented machine invented by Mr. Kondo. On an endless chain working horizontally is a series of small cast-iron pots. Into these the briquettes are placed and passed under a stream of matte. The iron cups are washed with lime water to prevent adhesion of the molten material.

The furnaces discharge into a square forehearth of wrought iron lined with blocks of liparite. The forehearth is divided in two unequal compartments by a water-jacketed partition, which does not reach to the bottom. The matte passes under this partition into the smaller compartment and runs into molds fitted on a small platform car, each car holding four molds. The lining of the forehearth lasts two months and the furnace crucible three months.

The charge for No. 4 furnace is 1,000 lb. of ore to which are added: Limestone, 230 lb., or 23 per cent.; converter slag, 200 lb., or 20 per cent.;

raw matte, 100 lb., or 10 per cent.; and coke, 130 lb., or 13 per cent. This furnace is used for pyritic smelting. About 50 charges are treated in 24hr., the capacity being 25 tons of raw ore or 45 tons including fluxes and coke. The coke consumption is 7 per cent. The matte contains 34 per cent. of copper and the yield or extraction is about 98 per cent. of the copper contents of the ore.

Furnace No. 2 is for roasted ores. The discharge into the forehearth is on the long side of the furnace. The air is led into a wind box made of the iron girders which support the upper portion of the furnace. The crucible is 14 in. deep. The slag from this furnace is not granulated, but made into plate slag to be used as flux. The capacity in roasted ore is about 28 tons. The charge is as follows:

	Lb.
Roasted ore	900
Raw matte from No. 4. furnace	413
Briquettes from roasted fines	413
Coke	130
Total	1,856

The capacity is 50 charges daily or a total of about 45 tons. The coke consumption with roasted ores is 13.3 per cent.

Furnace No 1 is the largest of the six. The air pressure is 30 mm. The air pipe around the furnace is 22 in. in diameter and main air conduit 36 in. in diameter. The depth of the crucible in front where the discharge takes place into the forehearth is 18 in. and at the back 12 in. Smelting was started with a forehearth on each end, but the matte fall was not sufficient to keep up a steady flow so the back forehearth was abandoned. The matte produced in this furnace contains 31 per cent. of copper. It is poured on to the floor from the pots, cooled by water and resmelted.

The coke consumption in this furnace on the total charge is 8.2 per cent. The charge is composed as follows:

	Lb.	
Raw ore	6,600	
Limestone	1,552	or 25 per cent. based on ore
Converter slag	1,321	or 20 per cent. based on ore
Matte	660	or 10 per cent. based on ore
Coke	925	or 9 per cent. based on ore
Total	11,058	

The capacity is 23 charges in 24 hr., 76 tons of ore or about 129 tons of mixture, being smelted daily.

Furnaces Nos. 5 *and* 6 have a capacity each of about 21 tons, ore and mixture. The spouts from the forehearths are made of blocks of liparite, covered with a hemispherical dome.

The cost per ton of concentrates and ore is estimated at 36.5 yen,

and as 7.63 tons of ore make one ton of copper, the mining and concentrating charges per ton of copper are 278.49 yen. The daily mining and concentrating expense would be 5,291.31 yen; and the cost per ton, based on the 528 tons raised, is 10.02 yen. The average value of the 528 tons of ore hoisted is 23 yen per ton. The cost of producing a long ton of metallic copper is given as 90.78 yen; the cost of mining and concentrating to produce a ton of copper as above, 278.49 yen or a total cost of 369.27 yen per ton, or about £37, but I think that these figures do not include general and administrative charges, nor the electrolytic refining, which is being done at Nikko, about 25 miles from the mine. The cost of coke is 26.70 yen per ton.

Bessemerizing in Converters

The converters are lined first with blocks of liparite, which are 8 in. square, and then decomposed liparite is plastered over them. The exterior diameter of the converters is 5 ft. The bottom lining is 12 in. thick and the top lining 7 in. The lining lasts 6 to 7 hr. and during that time 10 tons of matte is blown and as it contains 38 per cent. of copper, this should give 3.8 tons of copper. But there are produced at each blow only 2.9 tons of copper so that the rest, 0.9 ton, goes into the slag, skimmings, flue, etc.

For an output of 19 tons daily, there must be produced from the 146 tons of ore smelted daily, about 50 tons of matte. The matting furnace is circular. It has ten 3-in. tuyères and works at 22 mm. pressure. The crucible under the water jackets is on wheels and is tapped when matte is required for the converter. The charge in the furnace is: Matte, 660 lb; converter lining, 132 lb. or 20 per cent.; coke, 52 lb. or 8 per cent.

The remelting cupola smelts 50 tons of matte daily. The ladles for holding the copper are lined with powdered charcoal and clay (*Subai*). Slag is poured into these ladles to dry them. The ingots weigh 51 lb. A separate Root blower is used for the remelting cupola. In the converter the air pressure employed is 10 lb. per square inch. There are four converters in the building; while two are in use, two undergo repairs. The cost of bessemerizing, including smelting the matte, is 17 yen, £1 14s. per ton.

Cement Copper.—There are 10 large cement tanks through which the mine waters pass under similar conditions as explained for the Besshi mine. The cement copper is briquetted.

The water issuing from the water jackets is used for granulating the slag. The total quantity of water required by the six water jackets is 30 cu. ft. per minute. From the hollow box girders around the furnaces the iron bustle pipes connect with the tuyères.

The limestone used for fluxing purposes is quarried about 4 miles distant from the mines and the quarried material is sent down by an

aerial ropeway to a dumping place, where it is broken up by hand labor and then sent by horse traction on a tramline to the smelter.

The dimensions of the main flue are 6 ft. in width by 11 ft. in height. Before connecting with the chimney the smoke has to pass through a desulphurizing tower. The temperature of the gases and fumes is 200° C. before entering this tower, and on leaving only 87° C. This of course impedes the draft and a Guibal fan is placed before the tower to overcome this defect. In this tower a shower of lime water absorbs the sulphurous gases. The water passes through a filter after issuing from the tower so as not to vitiate the river water. About 25 per cent. of the sulphurous gases are absorbed.

The waste rock from the concentration works and the granulated slag are carried by aerial ropeways across the mountain range into another valley and there impounded by large dams, so that the storm water shall not wash the fines on to the rice fields below.

Analyses of Ores, Ashio Mine

	Lump Ore, Per Cent.	Grain Ore, Per Cent.	Fine Ore, Per Cent.	Pulp, Per Cent.	Cement Copper, Per Cent.
Copper	18.23	13.20	12.00	5.09	50.00
Iron	23.00	30.00	23.10	11.40	14.60
Sulphur	26.70	28.40	25.50	8.90	0.09
Silica	29.50	21.40	33.60	53 60	12.07
Aluminum	2.20	5.80	6.50	10.50	....
Lime	2.50	1.30	1.30	1.60	....
Arsenic	0.19	0.55	0.25	0.12	0.04
Zinc	0.33	0.65	0.63	1.06	0.06

Analyses of Mattes, Ashio Works

	Matte from Raw Ore	Matte from Roasted Ore
Copper, per cent	35.84	42.78
Iron, per cent	36.80	31.24
Sulphur, per cent	26.25	24.14
Arsenic, per cent	0.45	0.58
Gold, grams	1.00	1.00
Silver, ounces	30.00	30.00
Specific gravity 4.96	....	....

Analysis of Slag, Ashio Works

	Per Cent.		Per Cent.
Copper	0.36	Magnesia	1.71
Silica	39.80	Sulphur	1.30
FeO	35.67	Zinc	1.44
Al_2O_3	11.75	Specific gravity 3.51	....
Lime	11.57		

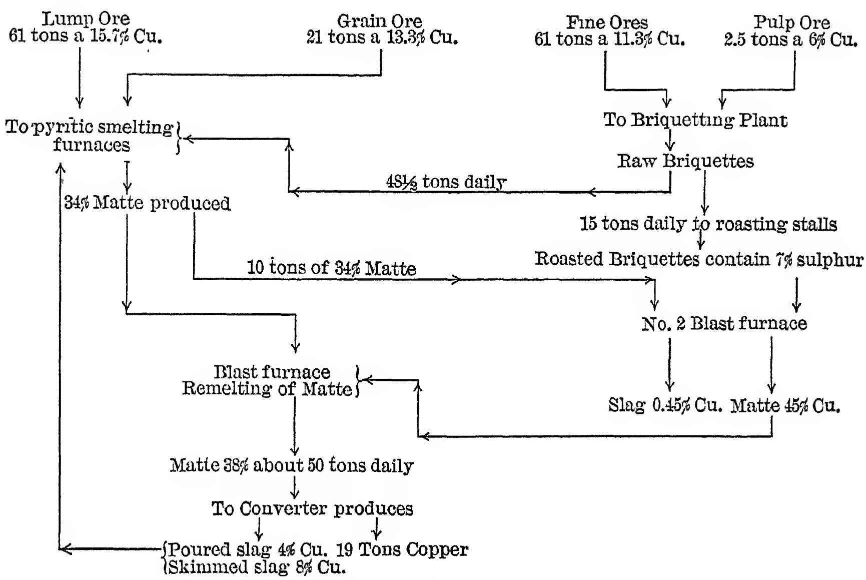

Fig. 3.—Scheme of Treatment at Ashio Smelting Works.

Analysis of Converter Copper, Ashio Works

Gold, dwt	0.208
Silver, oz	35.000
Nickel and cobalt, per cent	0.0249
Selenium, per cent	0.119
Tellurium, per cent	0.040

The percentage of copper is not stated.

Analysis of Speiss, Ashio Works

	Per Cent.
Copper	62.800
Iron	7.725
Arsenic	14.187
Antimony	0.488

In the bottom of the forehearth speiss collects occasionally, rich in gold and silver.

Analysis of Converter Slag, Ashio Works

	Per Cent.
Silica	19.86
Iron oxide	60.05
Copper	4.13
Alumina	7.81
Sulphur	1.26
Zinc	0.95
Magnesia	0.87
Lime	0.60

Analysis of Converter Lining, Decomposed Liparite, Ashio Works

	Per Cent.
Silica	74.47
Alumina	14.72
Lime	2.86
Magnesia	1.03
Other alkalies	1.31
Iron oxide	1.06

IV. The Kosaka Mine of the Fujita Company

The deposit of ore at the Kosaka mine is evidently of volcanic origin; it occurs near the junction of the tuffa and andesite. In its widest portion the orebody is 780 ft. wide and is proved for 2,000 ft. in length. The whole mass is mineralized.

A characteristic feature of this mine is that it contains three classes of ore. These form three distinct zones, and it seems as if nature had so prepared them that the metallurgist, by their proper commingling, should have his furnace charges without having to draw his fluxing materials from outside sources. All ores containing under 1 per cent. of copper are left standing in the mine as they are not considered valuable.

The black ore or *Kuromono* forms the eastern portion of the orebody and is the richest section of the mine. The silver and gold content increases with the rise in the percentage of copper in the ore. The vein material of this ore is chiefly heavy spar with small proportions of gypsum and quartz. The ore averages 2.5 per cent. of copper. The yellow or pyritic ore forms a band around the black ore on the western side and averages 2 per cent. when extracted for smelting. The largest part of the deposit on the western side is siliceous ore and when extracted averages 1.8 per cent. of copper. There is no zinc in this part, nor in the yellow ore, but the black ore contains over 13 per cent. of this metal.

In ancient times the Kosaka was worked as a silver mine and the following method of treatment was carried out there prior to 1868. The ore was broken into 1-in. cubes, and roasted with charcoal on a hearth about 2 ft. in diameter and 1.5 ft. deep. The latter was dug in the ground and lined with a mixture of slag and powdered charcoal. While roasting, a light blast from a Japanese bellows was turned on, which was increased

as the roasting advanced. Argentiferous lead collected in the bottom of the well and was ladled out. The small ingots thus produced were smelted on a shallow hearth made of wood ashes, a light blast turned on to oxidize the lead, which was removed as litharge; a silver slab remained behind. The matte and slag produced in the first operation, on account of the large percentage of zinc they contained, could not be utilized.

In 1870 Mr. Oshima introduced the following method:

1. The black ore was broken into 1-in. pieces and heap roasted.

2. The roasted ore and some earthy ore were mixed in certain proportions and smelted with charcoal, sand and slag in a blast furnace. The molten matte was ladled into a kettle and mixed with molten lead which had been smelted in another hearth. The mixture was well stirred and as the matte solidified the round disks were taken off. The argentiferous lead in the bottom was ladled out into molds.

3. The matte desilverized as above was broken into 2-in. pieces and roasted in heaps.

4. The roasted matte was smelted in another hearth furnace together with suitable proportions of litharge and slag and argentiferous lead regulus and a matte rich in copper obtained. This matte was roasted and smelted for crude copper, the resulting matte being again worked in the same manner.

5. The argentiferous lead regulus was treated by the English cupellation process.

Present-Day Metallurgical Treatment

There are smelted 1,200 to 1,300 tons of ore daily, or about 450,000 tons a year. There are produced 7,200 tons of copper per year showing an extraction of 1.66 per cent. of copper per ton of ore, therefore it takes 61.66 tons of ore to make a ton of copper. The smelting and refining charge is given as 4s. per ton of ore, or £12 0s. 8d. per ton of copper; the cost of mining at 3.2s. per ton of ore, or £9 1s. 8d. per ton of copper, making the total cost £21 2s. 4d. per ton of copper.

The smelting and mining charges seem small, especially since I am told that 12,000 people besides 400 officials are employed on the property. It is nevertheless quite possible that copper is being made here at about £35 per ton. The smelting cost for the first matte is given to me at 1s. 9d. per ton. The extraction on 2 per cent. ore is 83 per cent. There are seven smelting furnaces in this establishment.

1. Furnace, 60 ft. long by 3 ft. 7 in. wide, smelts 375 tons of ore daily.

2. Furnace, 19 ft. long by 4 ft. . wide,	smelt 875 tons of ore daily, or 175 tons per furnace.
3. Furnace, 25 ft. long by 4 ft. wide,	
4. Furnace, 25 ft. long by 4 ft. wide,	
5. Furnace, 25 ft. long by 4 ft. wide,	
6. Furnace, 25 ft. long by 3 ft. 4 in. wide,	

7. Furnace, 25 ft. long by 3 ft. 4 in. wide, smelts 110 tons of matte daily.

The blast pressure employed in the furnaces is 36 to 40 mm. The 25-ft. furnaces are 8 ft. high to flue bottom and 11 ft. 6 in. to charging floor. The 60-ft. furnace is 11 ft. high to flue bottom and 19 ft. 6 in. high to charging floor. There are eight Root blowers, operated by electricity, and requiring 400 h.p.

The furnaces discharge into semicircular forehearths of cast iron with a matte collecting trap, from which the matte discharges upon revolving tables. The 25-ft. furnaces have 15 6-in. tuyères on each side and two tuyères at the back. The wind pressure used is 35 mm. A series of dust chambers connects with the smoke stack, which is 200 ft. high and 16 ft. internal diameter. The diameter of the flues leading to dust chambers is 8 ft. The main air pipe is 6 ft. in diameter, and the branch pipes leading to these furnaces are 3 ft. 6 in. in diameter.

Bituminous coal to the extent of 2 per cent. is introduced through the tuyères. The 60-ft. furnace has 80 6-in. tuyères arranged in three tiers, so as to conform with the inside sloping bottom of the furnace.

Each of the 25-ft. furnaces requires 3,500 gal. of water per hour for cooling the jackets. The 60-ft. furnace is provided with a hydraulic ore-feeding device.

The mine ore after passing through grizzles is classified into lump ore and fines; the latter are hand sifted through 0.5-in. screens, the undersize being briquetted by means of wooden stamps. The daily production of briquettes is 150 tons. These are not furnace dried, but go direct to the smelter.

The mine ore is sent to the furnaces in the following proportions: Black ore, 5 tons; siliceous ore, 3 tons; yellow ore, 2 tons; total, 10 tons.

The siliceous ore is poor in gold and silver and contains 10 per cent. of sulphur. The average contents of the three ores are 24 per cent. of sulphur and 5 and 6 per cent. of zinc. As a metallurgical undertaking, I consider that the Kosaka mine is a difficult problem when the large percentage of baryta is considered. To the ability of Mr. Takeda and his able assistant, Mr. Takanuchi, is due the success of pyritic smelting introduced here by them.

The ore beds for the ore-smelting furnaces are composed as follows:

	Pounds
Black ore	4,300
Pyritic ore	1,000
Siliceous ore	2,700
Briquettes raw	2,700
Slags from second matte smelting	2,000
Total	12,700

The matte fall on an average is 9 per cent. of the quantity of ore

smelted or 112.5 tons per day, containing about 30 per cent. of copper. The first matte is reconcentrated in a special furnace, and the second matte contains 45 to 50 per cent. of copper.

The slags produced average 0.33 per cent. of copper. No coke or fuel is introduced into the furnaces at the feeding floor. The slags from the ore-smelting furnaces are granulated.

No. 7 furnace, which is used for matte concentration, is situated in a separate building. In the matte smelting 40 per cent. of siliceous mine ores and 1.5 per cent. of coke are added and 2.5 per cent. of bituminous coal is introduced through the tuyères. The slags produced here go back to the ore-smelting furnaces. The concentrated matte averages 50 per cent. of copper.

The second matte is crushed in rock breakers and then goes to Krom rolls which have a daily capacity of 30 tons. The pulverized material is roasted in Herreshoff furnaces down to about 4 per cent. of sulphur. There are 10 of these furnaces erected but only six are in operation. Each furnace has a daily capacity of 5 tons, so that they handle the 30 tons of second matte produced every day. Cord wood is used as fuel and 10 per cent. of the weight of ore roasted is used. The roasted ore is smelted in reverberatories to bottoms and white metal.

Smelting for White Metal.—For this purpose two reverberatory furnaces are employed. One 25 ft. long and 16 ft. wide, outside dimensions, the bridge plate of which is water jacketed; the other 20 ft. long and 12 ft. wide. The product here is white metal with 70 per cent. of copper and copper bottoms. It takes 4 hr. to melt a charge. The slag produced contains 4.87 per cent. of zinc.

Blister Copper.—From the two reverberatory furnaces the cold matte goes to the blister furnaces, of which there are two, and the blister copper produced goes to two refining furnaces. The copper from these is cast into anode plates which weigh 200 lb. The copper bottoms from No. 2 furnace are cast. In the same building are two furnaces for drying the mud or slime from the electrolytic work.

All the slags and skimmings produced in the reverberatories, liquation, blister furnaces, etc., go to the slag-smelting furnace.

Slag-smelting Furnace.—In the same building with the matte concentration furnace is a furnace for melting the slags produced in the reverberatory furnaces. This blast furnace is 10 ft. high and 3 ft. 4 in. wide, having a capacity of 65 tons.

In the slag-smelting furnace the charges are composed of: Slag, 100 lb.; scrap iron, 16 lb.; siliceous ores, containing gold, from Matsuoka mine, 13 lb.; coke, 15 to 20 per cent.

In this operation metallic lead collects in the bottom with a lead and copper matte on top; this separation is effected in a forehearth. The matte which is produced contains 10 per cent. of lead and copper. There

are produced 42 tons of lead monthly. The lead bars are liquated in five Japanese furnaces. The liquated lead after being enriched by the Parkes process is smelted in small reverberatories with the slimes produced in the electrolytic refinery. The bullion then goes to two English cupellation furnaces. Before the lead is cupelled, it is enriched by the Parkes process, two kettles being used for this purpose. The zinc employed is distilled in a special furnace with a retort attachment. The rich lead only is cupelled; the poor lead is sold. The lead-copper matte goes back to the matting furnace.

The electrolytic refinery is equipped with four continuous current generators, manufactured by the General Electric Co., of 220 h.p. each, operating at 75 volts and 2,000 amperes with a speed of 360; each machine is provided with an exciter. In the refinery are 500 tanks arranged in 10 rows of 50 tanks each. Each tank holds 20 anode plates, each of which weighs 200 lb., as stated.

It takes 30 days for the anode plates to dissolve and the daily production is 20 tons of cathode plates or 600 tons monthly. These plates weigh 20 lb. each. There is a special department for the purification of the electrolyte.

The Value of Ores and Production

	£	s.	d.
Average assay value, copper, 37 lb. at £60 per ton.	0	22	0
Average assay value, silver	0	6	0
Average assay value, gold	0	1	6
Total assay value of ore	1	9	6

The monthly production as given to me amounts to:

600 tons of copper at £60 per ton	£36,000
93,750 oz. of silver at 2s. per ounce	£ 9,375
750 oz. of gold	3,000
Total	£48,375

For the 37,000 tons smelted this would show a return of £1 6s. per ton, or a difference of 3s. 6d. per ton on the ore valuation. These figures also show that each ton of copper contains £20 worth of gold and silver and that the precious metals play a very important rôle in the profit earning capacity of this mine. The total costs per ton of ore treated are 11s. 4d. and the profit 14s. 8d. per ton, so far as I know the best on record in copper smelting. This does not include depreciation nor the cost of marketing the copper.

All machinery is driven by motors and in the various smelting departments 1,550 h.p. is employed. In the mine and in the other depart-

ments 1,450 h.p. is utilized. The generating station, which is 6 miles from the mine, develops over 3,000 h.p.

Analyses of Ores, Matte, Slags, Copper and Byproducts, Kosaka Mine

	Copper, Per Cent.	Lead, Per Cent.	Iron, Per Cent.	Zinc, Per Cent.	Silica, Per Cent.	Alumina, Per Cent.	Barium Sulphate, Per Cent.	Sulphur, Per Cent.	Bismuth, Per Cent.	Arsenic, Per Cent.	Antimony, Per Cent.
Black ore........	2.44	2.18	14.41	11.36	4.57	4.17	38.06	20.82	.....	......	....
Sulphide ore.....	3.21	0.58	27.69	5.20	10.59	4.08	14.30	33.00	.....	......	.....
Siliceous ore ...	1.72	0.17	18.69	1.20	49.01	4.90	4.51	18.92	.....	......	....
Briquettes.......	2.89	0.48	26.30	3.51	14.74	8.30	10.60	29.90	.....	......	.....
First matte......	30.35	5.48	28.03	8.18	0.95	0.53	0.11 sulphide	24.97	.....	......	.....
Ore slags.......	0.33	0.48	25.35	7.90	36.00	5.90 ..	......	0.99	.....	......	.....
Second matte....	49.83	7.57	12.04	4.00	0.50	0.50 ..	......	23 65	.....	......	.....
Matte slag.......	0.74	2.42	33.11	7.82	33.06	4.20 ..	......	0.64	.....	......	.....
Copper bottoms..	96.93	1.25	0.015	0.023	......	0.009 .. metallic	......	0.65	0.02	0.046	0.055
Anode copper....	98.96	0.106	0.003	trace	......	0.002 .. metallic	......	0.01	0.036	0.04	0.05
Electrolytic copper.	99.966	......	......	......	....	....	....	.....	0.003	0.0028	0.003
Electrolytic slimes.	4.085	11.837	0.156	0.870	5.610	3.230 ..	......	3.65	2.009	2.059	.....

The first matte from ore smelting contains 16.48 per cent. of barium oxide and the slag from matte smelting 5.8 per cent.

The specific gravity of black ore is 4.132; siliceous ore, 3.289; first matte, 4.762: first slags, 3.676.

V. Smelting at the Kano Works

The ore deposit at the Kano mine occurs in liparite and as far as present developments permit to judge, the ore formation took place near the line of contact with the schist rocks, which seem to form the hanging wall of the deposit. Therefore it is possible that future explorations may prove it to be a contact deposit. It is evident that the mineralization of the liparite took place by replacement or metasomaticism and that the orebody was not formed by the gradual filling in with mineral matter of a large cave or crevice. Subsequent earth movements exerted a crushing effect on the mineralized portion, which is evidenced by the large amount of brecciated material met with.

This mineralization took place along a wide zone, which, however,

narrows down in depth; reconcentration acting from the surface downward has been the cause of the formation of large lenses of enriched material.

The principal minerals found are iron pyrites, copper sulphides, blende, galena with some silver ores, whose nature has not been determined yet, and barite. The richest portion of the mine is that in which zinc predominates, forming lenses of black ore. In its main mineralogical features this deposit bears a close resemblance to the Kosaka mine, but the ore down to the 250-ft. level is not so rich. The open-cast workings show the orebody for 540 ft. in length and an average width of 300 ft., but numerous shallow pits prove that the orebody is over 1,000 ft. in length. In the absence of proper developments, however, no judgment can be formed as to the ore value in the unexplored portions. I consider that all ores over 1 per cent. in copper pay to work.

Owing to an insufficiency of iron in the surface ores, fluxing material has to be drawn from outside sources, but when the undecomposed or sulphide zone is reached, no doubt this defect will remedy itself and the mine ores will become self fluxing, as is the case with the Kosaka mine. Owing to the larger percentage of silica and earthy materials it has been possible to adopt, with the siliceous ores that predominate here, a system of concentration, which, in spite of the losses coincident with this process, effects a great saving as against the cost of direct smelting without concentration; the large quantity of iron required to slag the silica would more than likely make direct smelting prohibitive.

The present concentration plant shows a loss of 37 per cent., whereas when the new plant with better appliances is in operation this loss should be reduced to 30 per cent.

In regard to the smelting operations, I believe that the loss of about 14 per cent. in the smelting and refining department can be considered as normal, in view of the difficult character of ores dealt with, as they contain a much larger quantity of zinc than the Kosaka ores and at the latter mine after several years of experience a loss of 15 per cent. in the smelting and refining operations is acknowledged. On an average the ores going to the furnaces at Kano contain 12 per cent. of zinc, whereas at Kosaka they contain 7 per cent.

Considering that in the new plant 104,220 tons of ore will go to the smelter annually containing 12,506 tons of zinc, besides the zinc contained in the tailings from the concentration works, it is a matter of great regret that no process has been discovered by which at least a portion of this valuable metal can be recovered.

Ore Treatment

For 18 months, during the construction of the new plant, there was extracted daily from the mine 150 tons of ore. To the furnaces are sent

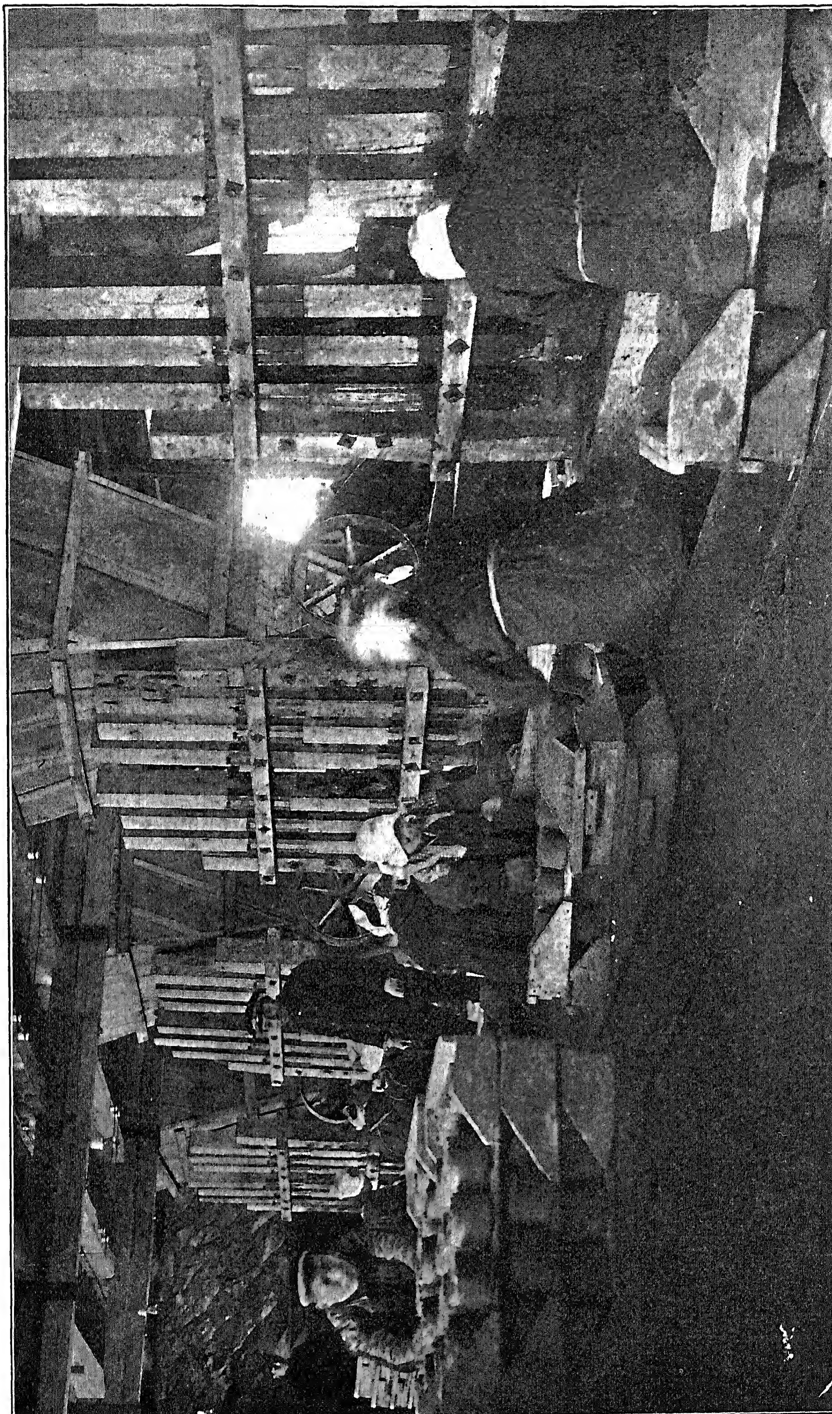

FIG. 4.—WOODEN STAMPS USED AT KANO WORKS FOR MAKING BRIQUETS.

direct 40 tons of lump ore and to the concentrators, 110 tons. The concentrator produces 30 tons of heading and 15 tons of middling making 45 tons of concentrate sent to the smelter. The total treated by the smelter is, therefore, 85 tons per day.

The concentrate is mixed in pug mills with clay ores from the mine, and by means of wooden stamps made into briquettes, as shown in Fig. 4.

As roasted pyrite from the sulphuric acid works is bought as a fluxing material, the fines are sifted out and added to the briquettes, together with the pyritic fines from the mine. There are 24 molding stamps.

The briquettes are dried in two furnaces, each having a capacity of 40 tons. Each furnace has three shelves. The flame from the fireplace passes over the first shelf, up a flue and back over the second, then through a flue in front and over the third shelf. The fuel consumption per furnace is 2 tons of a rather poor quality of bituminous coal.

The two furnaces are attended by 14 men working in 12-hour shifts or 28 in all per day.

It has been found that it is cheaper to utilize scrap iron than the burnt pyrites from the chemical works.

The present smelting works consists of two blast furnaces 9 ft. high by 3 ft. 4 in. wide, with six tuyères 6 in. in diameter on each of the long sides of the furnaces and forehearths in front of each furnace on the short side. From the tuyères to the bottom of flues the height is only 7 ft. The air main is 20 in. in diameter. The blast is furnished by two No. 3 and one No. 5 Root blowers run by a 40-h.p. dynamo.

From the forehearth the slag flows into pots to allow any matte particles in suspension to settle.

The matte concentration furnace is 40 in. in diameter and has six tuyères, 6 in. in diameter. The first matte produced contains 20 per cent. of copper and the concentrated matte 43 per cent. The slags from ore smelting are granulated. The ore bed in the first smelting is composed as follows: Briquettes, 826 lb.; mine ore, 495; slag from concentration furnace, 330; Mabuki slags, 82; scrap iron, 216; total 1,949 lb.

Each furnace smelts daily: Briquettes, 74,800 lb.; mine ore, 53,680; concentration furnace slag, 29,920; Mabuki slag, 7,480; scrap iron, 18,726; total, 184,606 lb.

The quantity of ore smelted in each furnace is about 50 tons and of fluxing material about 30 tons, making a total of 80 tons. The quantity of iron contained in these 80 tons is as follows: 50 tons of ore, 20 per cent. Fe, 10 tons; 17 tons of slag, 35 per cent. Fe, 6 tons; scrap iron, 8.5 tons; total, 24.5 tons. The ore and briquettes contain on an average 20 per cent. of iron, 21 per cent. of silica, and 12 per cent. of zinc. Air pressure at the furnaces is 26 mm.

It is well known that blende is decomposed by iron oxides and silicates, the resulting zinc oxide entering the slag; metallic iron liberates metallic

zinc, but most of the zinc sulphide entering the blast furnace remains undecomposed and enters the matte as well as the slag. Generally the percentage of zinc found by analysis in matte and slag will be about equal.

It will be seen from the analyses that both at Kosaka and Kano this rule holds good for the first matte and the first slag, but in the concentration furnace a greater proportion was driven into the slag, which is very important, as blende makes matte less fusible, obstructs proper settling and also carries other sulphides into the slags.

If zinc oxide has to be slagged off, its reduction to metallic zinc must be prevented, consequently the temperature in the furnace must not be too high, and the smelting must be done quickly, which no doubt accounts for the low ore columns in these blast furnaces; and a slag obtained not too high in silica, but rich in iron. In the presence of lime, it is customary to figure that one-half of the zinc oxide replaces one-half of the lime. The ore columns in these furnaces are 7.5 ft. in height.

It will also be seen that the slags contain about the maximum amount of zinc they should carry, namely up to 8 per cent.; if the percentage was higher the losses in metals would be large. If zinc oxide is reduced to metal in the lower part of the furnace by carbon, or by metallic iron, it becomes volatilized and forms accretions in the upper part of the furnace. For this reason an excess of iron must be avoided and the calculation of furnace charges carried out to a nicety. The zinc vapors will carry along lead and silver and becoming oxidized higher up, carry off metal as flue dust. If zinc oxide is not reduced to metal on its downward course in the furnace and then comes in contact with lead sulphide or sulphate, it is converted into sulphide in the presence of carbon. If sufficient iron is present, the iron will decompose the lead sulphide and the zinc oxide will remain unchanged.

I am not aware if ores similar to the Kano are at this moment being treated in the United States, and if not, these clever Japanese metallurgists have solved a problem in pyritic smelting which can be placed to their credit.

The matte fall in ore smelting is 10 per cent. of the quantity of material smelted.

The concentration furnace charges consist of:

	Lb.
First matte, 25 per cent. zinc	500
Mabuki slag	
Siliceous ores	235
Middlings from concentrator, 15 per cent. zinc	85

With the present plant the daily production of second matte is 8 tons.

An analysis of the products from the concentration works gave the following results:

	Jig Concentrate	Wilfley Tables
Silver, oz	4.00	4.00
Copper, per cent	3.22	3.59
Iron, per cent	19.65	21.09
Zinc, per cent	22.31	17.17

The Mabuki-Dokos

Four Mabuki-Doko hearths are used in connection with the matte concentration furnace as shown in Fig. 5. They are 3 ft. in diameter by 1 ft. 7 in. deep, each furnace holding 2.5 tons of matte.

The molten matte is poured into the Mabukis and wind pressure turned on to roast the metal for 4 hr. It takes 30 hr. to finish a charge during which time 675 lb. of charcoal are consumed or 12 per cent. of the weight of the charge. Each furnace produces 30 bars of black copper weighing 50 kins or 66.5 lb. (1 kin = 1.33 lb.)

During the first 16 months of smelting operations there were produced from 38,266 tons of ore: 479 tons of copper having a value, at £60 per ton, of £28,753; 3,812 mome of gold (1 mome = 10s.), £1,906; 442,636 mome of silver (1 mome = 15 sen), £6,639; total value, £37,298 This shows an extraction of 1.25 per cent. copper or 12s.5d.; gold, 1s.; silver 3s. 5d.; total value 16s. 10d. per ton of ore.

The assay value of the ore as coming from the mine is £1 5s.; consequently the loss in concentrating and smelting is 8s. 2d. per ton or 32 per cent. The gold and silver contained in a ton of copper produced formerly averaged £14, but in recent years the silver value shows an increase.

The mine is now being equipped with additional plant, so as to produce 200 tons of copper monthly. When this new plant is in operation, I estimate that the cost of making a ton of copper, including refining electrolytically and marketing will be £50.

The additional equipment in the new concentration plant includes: six Huntington mills, three Hancock jigs, 27 Overstrom tables, two Pindas concentrators.

The new smelting department comprises a smelting furnace 66 ft. in length, with two forehearths and two matting tables; two Root blowers to be run by a 150-h.p. electric motor; a chimney 150 ft. in height, with an inside diameter of 12 ft.; a new briquetting plant of 48 wooden stamps, together with four sets of ore-mixing machines and four drying furnaces; a Gates rock breaker, having a capacity of 300 tons in 12 hr.; and eight additional Mabuki-Dokos.

As the works are located in a highly cultivated country, filter presses were installed to clarify the water from the concentration works so as to avoid pollution of the river. The water is brought in a flume to a little valley and enters the presses under a pressure of 32 ft. In the settling

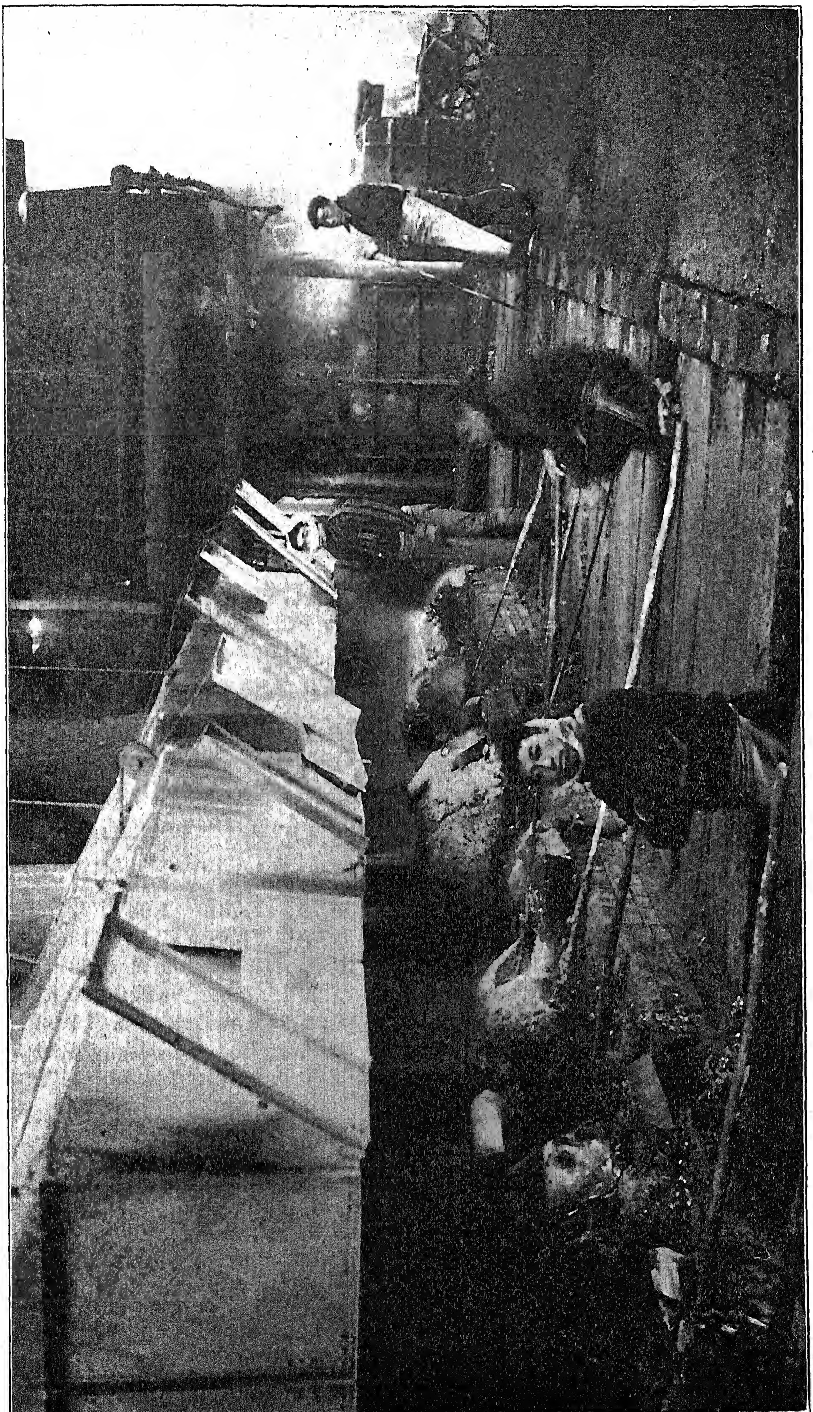

FIG. 5.—FOUR MABUKI-DOKO FURNACES USED AT KANO PLANTS FOR BESSEMERIZING.

ponds some powdered lime is added and the water is clear and drinkable on issuing from the presses. Each press takes 7 cu. ft. of water per minute and every five days the slimes have to be washed from the filtering cloth.

Experience has shown here that iron smoke stacks can be used for the smelting furnaces as a crust forms which protects the interior, whereas in the roasting furnaces, iron chimneys quickly corrode.

The power plant, which is being installed 3 miles distant from the mine, will furnish 500 h.p.

The fuel required in the metallurgical operations is as follows: Drying briquettes, coal, 8.33 per cent.; ore smelting, coal, 2 per cent., and coke 2 per cent. of the quantity smelted; matte concentration, coal, 8.8 per cent., and coke, 2.8 per cent. of the quantity concentrated; Mabuki-Doko, charcoal, 14 per cent. and coke, 2 per cent. of the quantity refined. When mining 16,850 tons of ore monthly, the cost of fuel per ton of ore will be (80 sen) 1s. 7d.

Analyses of Ores, Mattes and Slags at the Kano Mine

	Copper, Per Cent.	Lead, Per Cent.	Iron, Per Cent.	Zinc, Per Cent.	Silica, Per Cent.	Alumina, Per Cent.	Barium, Sulphate Per Cent.	Sulphur, Per Cent.
Black ore.........	2.37	3.34	10.50	20.94	27.74	6.98	2.18	17.40
Sulphide ore.....	2.27	1.07	19.74	12.82	19.70	5.78	3.98	22.26
Siliceous ore.......	1.36	0.37	9.97	7.81	52.30	7.42	1.46	13.87
Briquettes.......	2.47	1.11	17.00	13.16	22.53	8.16	3.21	22.50
First matte	19.95	1.18	37.24	9.82	1.50	....	....	26.62
First slag.........	0.39	......	32.75	8.32	36.00	1.56	....	1.89
Second matte......	42.94	1.21	21.36	6.81	0.86	....	....	24.18
Second slag..	0.54	......	34.53	8.68	31.30	6.64	....	1.73

The ore-smelting charges are made up of 53,680 lb. of mine ore and 74,800 lb. of briquettes or a total of 128,480 lb. Taking the average silica contents as 40 per cent., the charge contains 51,392 lb. of silica. The iron in the concentrates and Mabuki slag amounts in round numbers to 15,000 lb. and in the ore and briquettes to 18,000 lb., a total of 33,000 lb. This is not sufficient to slag such a difficult ore, and scrap iron to the amount of 18,726 lb. is added making the total amount of iron in the charge 51,726 lb.

VI. Smelting at the Tsubaki Mine

At the Tsubaki mine there is a large body of silver ore. This silver ore is smelted principally with copper ores purchased from outside mines. The orebody, which occurs in andesite, is divided by a shale dike 150 ft. wide; the northern orebody is called *Homa* and the southern, *Sozen*. The ore occurs in a crushed zone and looks like a mass of breccia which crumbles to pieces very easily; the ore carries lead, silver, sulphide and metallic silver.

The mine sends 116 tons of ore daily to the smelter. The fine ore is briquetted in a similar arrangement to the one described at Kano, 45 tons of briquettes being made daily.

The imported copper ores contain 4 per cent. of copper and 45 per cent. of sulphur. In the pyritic smelter 0.8 ton of Tsubaki ore is smelted to 1 ton of this 4 per cent. copper ore.

There are also brought here ores from the Karatoya mine of the following composition: Zinc, 28 per cent.; silver, 5.5 oz.; copper, 5.43 per cent.; iron, 12 per cent.; silica, 6.65 per cent.; BaO, 9.23 per cent.; and sulphur 29.31 per cent. The black zinc ores all carry barite in Japan.

Three smelting furnaces were in operation at the time of my visit but a new furnace 60 ft. in length was in course of construction.

The No. 1 furnace is 30 ft. in length by 40 in. in width and is used for reduction smelting; it has 20 tuyères on each of the long sides. The quantity of water required for the water jackets is 25 cu. ft. per minute. The daily ore and material smelted amounts to 173 tons in the following proportions:

	Tons	Parts
Tsubaki silver ore	75	100
Purple ore from acid works	34	45
Limestone	23	30
First matte from same furnace	23	30
Sulphide ore	3	4
Coke	15	20
Total	173	229

The so-called purple ore is the residue from roasted pyrite furnished by the acid works.

The daily matte production is 40 tons which is re-treated in the same furnace and in the matte concentration furnace. The matte produced contains 20 per cent. of copper.

The No. 2 furnace, which is used for pyritic smelting, is 9 ft. long by 3 ft. 3 in. wide. The furnace has a forehearth with two matte siphon taps. The quantity of ore smelted here is 60 tons daily and the charge consists of:

	Tons
Tsubaki silver ore	30.0
Sulphide ores, outside mines	30.0
Zinc ores	9.0
Metallic iron (scraps)	7.5
Slag from concentration furnace	12.0
Coke	1.5
First matte	10.0
Total	100

There is introduced through the tuyères 2.5 per cent. of bituminous coal. The cost of pyritic smelting in this furnace is given at 5s 6d. per ton. The matte, of which 10 tons is produced, contains 18 per cent. of copper. This matte is sent to the concentration furnace.

The height of the matte concentration furnace from tuyères to the flue is 8 ft. The charge consists of:

	Tons
First matte from reducing and pyritic smelting	20.0
Second matte, circulating	4.0
Slags from Yamashita and Mabuki	5.0
Tsubaki silver ore	11.0
Limestone	2.0
Coke	0.5

This furnace produces 10 tons of concentrated matte containing about 39 per cent. of copper; the slag contains considerable iron and 3 per cent. of copper and is sent back to the pyritic smelter. The cost of smelting in this furnace is given as 4s. 6d.

The blast is furnished by two Root blowers operated by a Pelton wheel. The pressure on the Pelton is 110 ft.; volume of water 15 cu. ft. per second; diameter of water pipe 30 in.; power developed 140 h.p. The two blowers require 85 h.p., and give a wind pressure of 30 mm. The remaining 55 h.p. operates a dynamo furnishing power for the locomotives on the electric railway. The main air pipe is 40 in. in diameter; branch air pipe to big furnace 30 in. The pyritic furnace has six tuyères on each side, the air pipes around the furnace being 12 in. in diameter.

First Matte Desilverized in Lead Bath

The matte produced is drawn from the forehearth, as shown in Fig. 6, into a cavity, lined with brasque and covered with a dome of fire clay, in the bottom of which is 0.5 ton of molten lead. The lead bars which are placed in this cavity are produced from litharge, which results from the cupellation process. After some lead bars are introduced and melted by the hot matte which covers them, more bars are introduced until the requisite quantity is completed. A log of wood is then placed on top of the bath and pushed into the bottom by means of an iron bar introduced through a hole in the cover and kept in position by a cross

piece held down by two men. This causes an active ebollution in the bath, and a thorough stirring of the lead with the matte. The bath is then allowed to settle and the matte, containing 45 per cent. of iron and some lead, drawn off and put back into the furnace. The excess matte goes to the other two furnaces. After drawing the matte, the lead bath is skimmed clean and the metal ladled into molds.

To keep the bath in a fluid condition a few pieces of wood and charcoal are kept burning on top of it. There are two cavities, one on each side of the forehearth, which are used alternately for treating the matte as

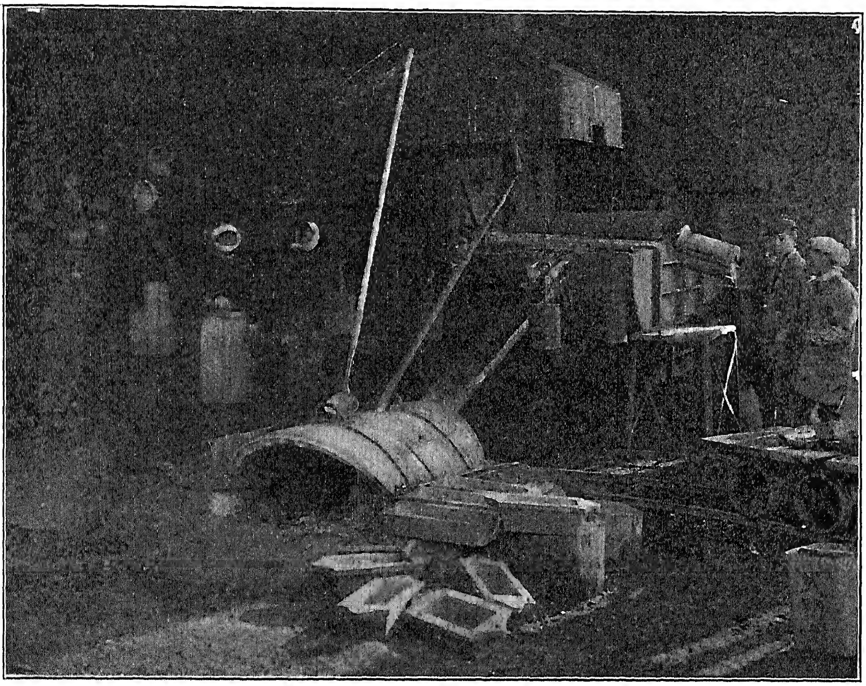

Fig. 6.—Smelting Furnace at Tsubaki Works, Showing Matte-Desilverizing Well with Dome-shaped Cover.

above described. The lead bars produced here contain 570 oz. of silver to the ton and as the lead is impure it is first sent to the liquation furnace.

The "smelting in" or desilverization of matte (in German, *das Eintränken*) in a metallic lead bath is an old process and has for its object the decomposition of the silver sulphide and the absorption of the liberated silver by the lead. As the Tsubaki ores contain a certain proportion of metallic silver, it is hardly to be supposed that any would remain in that state after passing through the furnaces. As silver sulphide is not decomposed completely by the lead in one operation, the "smelting in" has to be repeated, and it is only after several operations that the complete desilverization of the matte is accomplished.

In some establishments the matte is smelted first in a reverberatory furnace and lead added and stirred in. In that case one part of lead is added to three parts of matte. After thorough stirring, the matte is allowed to settle and as it cools it is lifted off in disks; the lead, owing to its greater specific gravity, separates very nicely in the bottom of the hearth. The hotter the furnace is kept the more complete the desilverizing action. The matte is then re-treated in the blast furnace.

The operation is made more complete if in the molten matte and lead bath a log of wood is introduced and by means of iron bars is held on the bottom of the hearth and kept there for several minutes, as the gases developed from the wood bring all of the molten and metallic particles

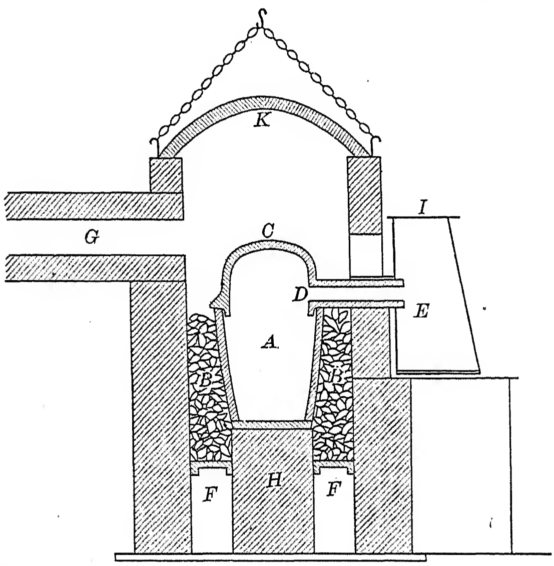

FIG. 7.—ZINC DISTILLING FURNACE AT TSUBAKI PLANT.

into intimate contact. After the matte is drawn, the lead can be ladled out or it can be tapped. When matte is treated in this way in small reverberatories it is necessary to deal with the same matte at least four times, as just explained, to effect its complete desilverization.

The lead produced in the first operation may be rich enough to go direct to the cupellation furnace. After the fourth operation the matte now poor in silver goes to the ore-smelting furnace.

In Tsubaki the matte after treatment contains 45 per cent. of iron. The lead ingots are taken now to the liquation furnace.

The liquation of these ingots is effected on a cast-iron plate built in on top of a brick square. This plate is turned up to form a rim 3 in. high and in the front a spout is provided. Hot charcoal is used and the ingots placed on top. The iron plate has an inclination of 12° and the molten lead passes out through the spout, collects in a little cavity in the

ground and is cast into small square ingots which are sent to the cupelling hearth. The dross which remains goes to the open-hearth furnace or the Yamashita Buki.

In this process the largest portion of the silver and lead which is in alloy with the copper is smelted out, leaving a skeleton of copper behind with some silver and lead, whereas a small proportion of copper liquates with the lead. So far as I know this process, known in Germany as the *Saigerprocess*, is no longer in use there.

The matte from the concentration furnaces goes to the five Mabukis in a separate building. Each Mabuki holds 3 tons of concentrated matte, containing 40 per cent. or 1.2 tons of copper, per charge. Coke is used as fuel. There is added in the Mabukis 7 to 8 per cent. of litharge. The black copper produced here goes to special liquation furnaces where the lead is separated out. It is interesting to note that the black copper produced contains 3 per cent. of silver, two-thirds of which liquates out with the lead while one-third remains in the copper. The time of treating one charge in the Mabukis is 30 hr.

The Japanese *Nanban-Buki*, or liquation furnace, used for treating the black copper ingots, is a very small affair, being only 1 ft. 6 in. deep, about 2 ft. wide and 1 ft. 6 in. high. The four furnaces in use are built of brick lined with tiles of fireclay, and are provided with iron smokestacks which connect with the main flue of sheet iron leading to the dust chamber. Coke and charcoal are used as fuel. The blast at 10 mm. pressure is introduced through the top. The air upon striking the back wall is deflected upward and then takes a downward course to the front opening. Coke and charcoal are used as fuel.

Each shift of 8 hr. works up 0.25 ton of black copper ingots in each furnace. When the ingots become red hot and soften, the workman presses them with a piece of wood fastened to an iron rod. A charge consists of three ingots weighting 200 lb. The speiss which collects separately is pressed and passed on to the cupellation furnaces. The Mabuki black copper which is treated in these liquation furnaces contains: Lead, 4 to 5 per cent.; silver, 3 per cent.; copper 89 per cent.

The lead produced here goes to the cupellation furnaces and contains 10 to 13 per cent. of silver. The speiss contains 35 to 40 per cent. of silver. The black copper which remains behind is cooled in the furnace with a spray of water, and is then taken out and broken with a sledge hammer.

Black Copper Smelted, in Open-Hearth Furnaces

The Yamashitas, or open-hearth furnaces, like the Mabukis, are built in the ground. The blast is introduced at the back. Coke and charcoal are first burned to heat the cavity and then the copper is flowed into a ladle suspended on a chain from which it is poured into molds on a moving platform car underneath. This copper contains 0.5 per cent. of lead.

The copper from the Yamashitas assays 96.5 per cent. of copper and 1 per cent. of silver. The iron ingot molds are heated, by burning charcoal in them before the metal is poured.

The cupelling is done in two furnaces. The cupelling hearths are made of cement and can treat 2.8 tons of work lead in one furnace in 24 hr. The loss in cupelling is 7 per cent. The litharge as it flows from the cupel passes through incandescent charcoal placed in a cylindrical stove and is reduced to metallic lead, which collects in a cavity in the ground and is cast in molds. The pressure for the blast is 3 to 5 mm. The antimony speiss from the liquation furnaces is added during cupelling. The quantity of metallic silver obtained here is 51 kg., and the monthly production averages 1,600 kg. The silver contained in the black copper amounts to 480 kg. making a total of 2,080 kg., or in round numbers 70,000 oz. of silver. If all the silver came from the 3,480 tons of Tsubaki mine ore this would be an average of 20 oz. to the ton, but there are no data available on the subject.

A new plant is in course of erection in which the concentrates will be briquetted and the tailings lixiviated by some process to be determined after experiments are finished.

Analyses of Ore, Slag, Matte and Furnace Products, Tsubaki Mine

	Silver Oz.	Lead Per Cent.	Iron Per Cent.	Zinc Per Cent.	Silica Per Cent	Alumina Per Cent.	Barium Per Cent.	Copper Per Cent.	Sulphur Per Cent.
Homa silver ore	33.0	1.0	2.5	4.0	29.5	4.5	43.3	20.00	......
Sosen silver ore	31.0	trace	4.5	2.0	40.0	2.2	30.0	....	25.0
Matte, reducing smelting	187.0	7.0	36.0	2.0	......	.....	......	....	......
Slags, reducing smelting	1.4	trace	25.0	2.5	37 0	4.5	17.5	0.20	......
Slags, pyritic smelting	1.8	trace	22.0	3.5	38.0	5.0	14.0	0.12	......
Matte, pyritic smelting	117.0	5.0	38.0	3.0	......	.....	......	18.50	......
Concentrated matte	503.0	4.0	29.3	1 2	......	.....	......	38.80	23.7
Black copper from Yamashita	400.0	0.5	......	.....	......	.....	......	96.70	......
Slag from Mabuki	6.0	2.3	32.0	.....	8.0	5.0	0.7	15.90	8.0
Concentration slag to pyritic smelter	2.0	.....	28.0	.....	34.5	7.5	10.0	0.30	......

The Parkes Process in Use

As a certain portion of the lead which is produced in the matte-desilverizing wells is not rich enough in silver to go direct to the cupellation furnaces, it is submitted to the Parkes process.

The lead which has to undergo the Parkes treatment is melted in cast-iron kettles built in over fire places. Zinc is then added in portions of 1 to 2 per cent. With lead poor in silver two additions of zinc only are necessary, but with lead rich in silver and also gold, three, four or five additions may become necessary. It is necessary to employ pure zinc for this operation. When the first portion of zinc is added it absorbs any copper and gold which is in the lead bath; and there is a great advantage

in this, as the further additions of zinc when collected are free of copper. The zinc after it is introduced in the lead bath is melted by raising the temperature of the lead bath to the smelting temperature of the zinc, and the contents of the kettle are stirred. The lead bath is then slowly cooled, and as the zinc, lead, silver, gold and copper scum rises to the surface it is carefully skimmed with a perforated ladle. When it is noticed that the lead begins to freeze, the temperature is raised again and a second portion of zinc added, the same operation being repeated three or four times until the assay shows that there are only 1 or 2 dwt. of silver left per ton of lead.

If there is any copper in the lead it goes over with the silver into the zinc. Any gold in the lead goes over into the first zinc which is added and forms a zinc-gold crust with a little silver. Nickel and cobalt are also absorbed by the zinc. Antimony remains in the lead; if present in large proportion it retains silver in the lead; if present in quantities not exceeding 0.7 per cent. it does not interfere with the zinc process. Arsenic and tin remain in the lead. Arsenic delays the desilverization and prevents a good separation of the zinc crust. Bismuth remains in the lead. Tellurium, platinum and palladium go into the zinc. Therefore if lead contains these impurities it should be purified before going to the zinc process.

The zinc scum obtained from the above process is liquated. This operation has for its object the removal of a portion of the lead, as the melting temperature of the lead is lower than that of the scum. If the zinc scum is heated to the melting temperature of the lead, the lead will liquate out, but the temperature must not be so high as to cause the oxidation of the lead and zinc, as a sort of a mush or pasty alloy will be formed which is infusible and resists further separation. After separating a portion of the lead, the remainder is called rich scum. This liquation is generally carried out in kettles, each provided with an inclined bottom and a spout so that the liquated lead can run out and collect in a sump to be submitted anew to the desilverizing process.

The furnace employed at Tsubaki for treating the rich scum is shown in Fig. 7. Graphite crucibles *A* are used which are placed in a round wind furnace, *B*. The covering consists of a dome-shaped hood, *C*, provided with an outlet, *D*, which connects with a condenser, *E*, made of cast iron. The grate bars are at *F* and the flue is at *G*. The crucible is placed on the brick foundation, *H*. The heating is done by means of coke. The cover is of fire clay and can be raised and lowered by chain and counter weights.

The scum is heated in the retort; the zinc evaporates and is collected in the condenser as oxide and metal, whereas the lead and silver remain in the retort.

The rich scum is mixed with 1 per cent. of charcoal powder and introduced in portions of about 500 lb. into the crucible. After the dome is

luted on with fire clay and the pipe *D*, inserted and also luted, coke is placed around the crucible and glowing coals are placed on top. As soon as gases begin to escape from the pipe *D*, the cover is placed on the condenser.

After the operation is finished the condenser contains metallic zinc, about 30 per cent. of the charge, and zinc dust, about 8 per cent. The residue in the crucible, composed of lead, precious metals and copper, goes to the cupellation hearth. The coke consumption is about 80 per cent. of the weight of the charge.

VII. Old Methods of Treating and Smelting Ores

The Crushing and Pulverizing of Ores.—A stone wall was built, 3 ft. square and 3 ft. high, in the center of which a square block of granite or other hard rock was placed and well tamped in. Upon this rock the ore was broken with heavy hammers and the pulverized material passed through horse-hair sieves, 40-mesh to the square inch, or through bamboo sieves.

Concentration.—The pulverized ore was treated in dolly tubs, the portion which floated being passed through launders while that which settled was further concentrated by vanning troughs. The tailings collected from vanning were ground in stone mortars and washed again, and the slimes were made to flow through a launder named *Neko*. The Neko is a frame over which a coarse cloth is stretched, about 10 ft. long, set obliquely. The portion of the slimes which remained on the cloth was again treated in dolly tubs and vanning troughs.

Granulated or grain ore was put into shallow willow baskets *Zaru-age*, where it was subjected to a jigging motion in a basin of water. The fine particles which escaped through the meshes of the sieve were caught in the basin and passed over the *Neko;* that which remained was panned in a dish called "*Yurishita.*"

Roasting the Concentrated Ore.—The concentrates were mixed with clay and made into balls, which were heaped on to a charcoal fire in hearths made of earth. The sulphur was burned and the minerals oxidized.

Smelting in Open-Hearth Furnaces, Do-Buki

This furnace is simply a cavity in the ground, of hemispherical form, with a diameter of 0.5 to 1.5 ft., and lined with brasque. The blast is supplied by an ordinary smith's bellows called "*Fuigo,*" a square box in which moves a piston packed with a badger's skin. The piston rod is moved backward and forward by a coolie, who draws it out with the hand and pushes it back with the foot. Usually two such bellows are used with each furnace. The bellows furnish 4 cu. ft. of air per piston stroke or about 120 cu. ft. per minute.

The tuyères open into the upper border of the furnaces. The direct ascent of the blast is prevented by a vaulted roof of clay which extends above the orifices of the tuyères as far as half way across the furnace. The furnaces are separated from the bellows by a back wall, the products of combustion escaping through a chimney, built of framework and covered with loam, which is supported partly by the back wall, partly by pillars and commences about 7 ft. above the bottom of the furnace.

To blow in, the furnace is filled with charcoal, a fire lighted and the blast turned on lightly. Charges of ore are thrown in and the blast increased until complete fusion takes place.

When the furnace is filled with molten material nearly to the tuyère level, the blast is stopped, the burning charcoal raked out and water sprinkled over the fluid slag. As the slag solidifies, it is removed down to the fluid matte; then charcoal and ore charges are added and the operation repeated until the furnace is filled with molten matte and metal, whereupon the contents are ladled out or removed in disks, as the matte gradually cools. The red hot furnace is then repaired with clay and smelting operations resumed. When the furnace becomes unfit for use, it is cooled down, repaired with brasque, warmed over night and is ready for operation again the next morning.

In such a furnace from 3,500 to 4,000 lb. of ore can be smelted in a day, and some mines have a number of them in operation. The treatment in these furnaces is a very wasteful operation and the fuel consumption ranges from 30 to 70 per cent. of the weight of the ore. When ores are smelted containing gold, a certain proportion of metallic lead is smelted in and the bottom collects auriferous lead and matte.

Roasting the Matte.—The matte is roasted in stalls built up of common stones, of dimension suitable to local conditions. Brush wood or split cord wood is placed on the floor of the stalls, the matte piled on top of it, care being taken to leave vent holes, the pile covered with fine ore or fine matte and the fuel ignited at the air hole. The time required for roasting the matte depends on the size of the roast heap. Pieces of well-roasted matte are selected for the next operation, treatment in the Mabuki-Doko.

Fusion of the Calcined Matte in the Mabuki-Doko.—About 0.5 to 1 ton of the roasted matte is melted down in a hearth furnace similar in construction to the one above described. When all the charges have been melted down, the slag is removed and the front half of the top of the hearth which remained uncovered, is covered with a heavy piece of fireproof clay tile, supported on short pillars of the same material.

A small opening is left in the front of the cover and an auxiliary tuyère inserted so that the blast is directed on the surface of the molten bath. When the smelting is completed the cover is taken off and the slag scraped away. The slag still remaining is solidified by sprinkling water on the surface. This slag is resmelted in the ore furnace. The

matte thus exposed is removed in thin disks after cooling. The whole operation takes about 10 hr. with a consumption of charcoal amounting to 25 to 35 per cent. of the weight of the charge.

When precious metals are to be won, lead is smelted in with the copper in the bottom of the furnace, making an alloy of precious metals, lead and copper; this operation is called *Awase-Buki*, and is sometimes carried out in a separate hearth furnace.

Black copper containing more than 30 oz. of silver per ton is melted with about 40 per cent. of argentiferous lead in a hearth furnace with one tuyère which inclines downward. The charcoal is kindled and 270 lb. of black copper is melted down. About 70 to 80 lb. of lead are then added. When the hot coal, slag, and dross are raked out from the hearth, an iron bar with a spherical knob is dipped into the molten metallic bath, on which the metal solidifies and adheres, forming a thick crust. The bar is now removed, the incrusted end immersed in cold water and the shell or crust knocked off by a hammer. This is repeated until all the metal is removed from the hearth. Six or seven charges are operated daily in the same furnace, with a consumption of about 20 per cent. of charcoal.

Liquation in Namban-Buki, or Namban-Shibori.—This process has for its object the removal of the lead from the copper, silver and lead alloy, and in this manner most of the silver leaves the copper. This is accomplished by heating the black copper in small furnaces with a light blast to a temperature sufficiently high to melt lead but not copper. The molten argentiferous lead is drawn off leaving the copper skeleton behind. As this copper still contains some silver it is smelted again with lead, during the second liquation becoming very much impoverished in precious metals.

In one of the Osaka refineries the liquation furnace has an oval form and is covered with loam on the outside and lined with brasque on the inside. The top of the furnace is covered with a tile, leaving only a small semicircular opening in front, which is also covered with a tile. This opening is used as a charging hole for the charcoal during the operation. The open front of the furnace is also closed by a movable tile which does not reach to the bottom, but leaves a narrow space for the working of the metal.

An iron blast pipe directs the blast downward. The inclined working floor in front, formed by ramming down brasque between the side stones, terminates in a shallow cavity for receiving the liquated lead.

About 116 lb. of the alloy are charged in lumps, charcoal is piled in, the tile covers put in place, and the blast turned on.

When lead begins to flow out, and the mass becomes pasty, it is squeezed with a wooden rod, fastened to an iron handle; when it hardens it is pushed back into the furnace, so that it may become reheated. The process is repeated until almost all the lead is liquated out, the operation lasting 2.5 hr., with a consumption of about 50 lb. of charcoal.

The quantity of copper alloy which is treated in one day in one furnace is 350 lb.

Smelting the Liquated Copper.—After liquation the copper is subjected to an oxidizing smelting in circular open-hearth furnaces. Charges of 275 lb. are smelted with charcoal and the blast, directed on to the surface of the bath, is turned on. In about 1.5 hr. the metal fuses, whereupon the hot charcoal and slag are drawn out and the surface of the metal bath is blown by the blast causing the phenomenon called copper rain; this lasts about 10 minutes and the mass is then allowed to cool gradually, when the frozen copper is removed in thin disks.

The whole operation lasts 2.5 hr. with a consumption of 165 lb. of charcoal. About 1,500 lb. of copper residues are smelted daily per hearth, producing 1,470 lb. of rosette copper and 33 lb. of slag, which contains 10 per cent. of copper.

Cupellation, Hai-Buki.—In some of the old methods this operation is carried out without a blast, by placing the cupel in a sort of a muffle furnace, but in most cases the cupel is placed on a hearth and a blast turned on from a bellows.

This is generally effected on a small open hearth, constructed of a wooden box, 3 ft. square and 2.5 ft. deep set in the ground. The cupel is made of wood ashes, the soluble salts being leached out.

In the center of the cupel is a circular cavity on which a charcoal fire is made and 100 lb. of lead added. The charge is then surrounded with tiles 1 ft. square and the blast turned on. As the metal fuses the charcoal is removed to the periphery and the blast made to play slowly on the metal bath. The litharge formed swims on the top of the bath and is absorbed by the ashes. The operation lasts about 2.5 hr., consuming 20 lb. of charcoal.

At a gold mine which I visited, the concentrates of arsenical pyrites, containing some galena, are briquetted by hand labor with calcined feldspar. After air drying they are smelted down in an open-hearth furnace, the blast being furnished by a small fan. The slabs of matte obtained in this first smelting are resmelted in a smaller open-hearth furnace and the matte, or rather speiss, taken off in thin slabs. During the second smelting a pool of lead collects in the bottom of the furnace, which is ladled out into a mold and cupelled in the manner above described giving gold bullion about 700 fine.

Refining Silver Bullion, Parting.—Silver bullion is fused on a flat hearth, and lead and sulphur added to convert the silver into a sulphide. The silver matte thus produced is sprinkled with water and cooled, or it is poured off carefully while still liquid, leaving the gold on the hearth.

The silver matte thus produced is again melted with lead, until all the precious metal contained in it is absorbed by the lead. It is then heated strongly to burn away the sulphur, and more lead is added; when the operation is finished, the lead is ladled into a mold and then cupelled.

Any silver and gold produced must come up to the standard of the *Hoji Koban.*

Toughening the Copper.—This operation is performed in clay crucibles heated in a hearth lined with brasque; charcoal is packed around the crucible, and a clay tuyère is fixed with an inclination of 20° to play on top of the metallic bath. Charcoal is then piled on top and the crucible heated to redness after which 66 lb. of copper are introduced, covered with charcoal and the blast turned on. In about 25 minutes, the charge completely melts down, and the slag and dross are removed. The molten metal is stirred well with a long hard stick of charcoal in order to toughen the metal or to reduce the cuprous oxide in the molten mass, an operation which is completed in a few minutes; the metal is then cast in molds.

Discussion

J. W. Richards, So. Bethlehem, Pa.—I do not see in the paper a description of the Mabuki hearth as used for ore smelting. I noticed it described for concentrating matte, but I believe the older Japanese process was to use it for smelting directly, by a pyritic smelting and Bessemerizing all in one operation, in the same simple hearth apparatus. There is a monograph published in German, on Copper-Ore Smelting in the Mabuki Hearth.

At Ashio the practice of injecting bituminous coal at the tuyères is noteworthy. It is very interesting to see two men load little cartridges on the end of an iron rod with bituminous coal, and push them into the furnace tuyères, and with a piston inject 2 or 3 lb. of bituminous coal into the smelting zone of the furnace. Moreover, the statement was made to us that without the extra heat generated directly in the smelting zone by this bituminous coal, the furnace did not operate well.

In the further description, at the Ashio works, it is stated that copper is deposited from the mine waters by means of iron. When we visited at Ashio in 1911 the methods had been changed, since the solution flowing from the precipitating plant was destroying the rice fields in the valley. A method of treatment by fractional precipitation by lime was then in use, which I described in a small paper in the Institute *Transactions* for 1912.

At the refining works at Nikko there was an interesting refining plant on the series system, which had been put up by a Japanese who came to this country, and worked at Baltimore, and carried the process *en bloc* to Nikko.

The tradition, however, was faint and the practice was poor. I have heard within the last week that a new plant is being erected to replace this inadequate one at Nikko, using the series system

Coal-Dust Fired Reverberatories at Washoe Reduction Works

BY LOUIS V. BENDER, ANACONDA, MONT.

(New York Meeting, February, 1915)

After investigating the work of coal-dust fired reverberatories of the Canadian Copper Co., at Copper Cliff, Ontario, the management of the Washoe Reduction Works decided to experiment with and ascertain the advantages of using coal dust as a fuel in their reverberatories. Consequently, during the month of June, 1914, reverberatory furnace No. 8 was changed in order to use powdered coal as a fuel. The results obtained by this method of firing are gratifying and show a decided saving in cost of smelting as compared to grate firing with lump coal.

The furnace, as remodeled, is 124 ft. long by 21 ft. wide, and varies in height from 8 ft. 6 in. at the back end to 5 ft. 7 in. at the skimming end. The general construction of the furnace is similar to that of the other furnaces at this plant. There are no side doors to this furnace, as it was thought that with the present arrangement for feeding no "fettling" or "claying" would be required. The interior of the furnace can be inspected through the burner port holes, after shutting off the burners and giving a few seconds' time for the gases inside the furnace to clear away. The charging is done on either side of the furnace from longitudinal hoppers, extending a distance of 74 ft. from the back end of the furnace. Leading from the hoppers into the furnace are 6-in. pipes, spaced 19¼ in. apart, through which the charge is intermittently dropped. The charge is kept well above the slag line at all times; in this way the side walls are protected and no fettling is needed on this portion of the furnace. The remainder of the furnace requires fettling. After operating for three months, we found that the bricks were eaten into along each side wall from the skimming door back to the point where the charge had been dropped. The depth of this cutting away was 8 in. close to the front end and gradually tapered to zero at a distance of 50 ft., and was greatest on the side of the furnace having the larger flue connection. Hoppers will be put in for the entire length of furnace, from which fettling material will be dropped, to prevent this cutting.

After the run of three months, the roof was in excellent condition. At the back of the furnace the bricks were not cut into at all; at 30 ft.

from the back end they were eaten away 2 in., but at 60 ft. distant they were as when put in. The roof is 20 in. thick. After operating for a while trouble was encountered in tapping the matte. The tap hole was on the east side of the furnace 83½ ft. from the front end. Charging could not be done over the tap hole, or for a distance of several feet on either side; also, owing to the method of charging, matte accumulated in the front of the furnace and could not be completely drained through the side tap hole.

When the furnace was down for fettling in front, it was seen that the calcines fed into the furnace sloped very gently from either side to the center. This, of course, took up the space which in other furnaces is filled with matte and forced the matte to the front of the furnace, and also prevented its being drawn out at the side tap hole. The furnace will not hold more than 50 tons of matte; the other furnaces hold 175 tons. Finally it was decided to tap the furnace at the front. A suitable runway was put in and a tap hole made to the side of and below the skimming door, and all of the matte was tapped therefrom. About 35 tons of matte is tapped per shift. The furnace is skimmed three times per shift. The gases are taken from the furnace through brick flues to either of two batteries of Stirling boilers, each battery developing 650 h.p. One of these flue connections was left as before with a cross-sectional area of 13½ sq. ft., the other flue connection has a cross-sectional area of 40 sq. ft. The smaller flue connection is used only whenever it is necessary to clean the boilers connected with the larger flue. This occurs once a month and lasts for a period of three days, during which time the tonnage smelted is considerably less than when using the larger flue. The following figures verify this statement.

Cross-Section of Flue	Average of	Tons	Ratio
13½ sq. ft.	3 days	405	6.8
40 sq. ft.	3 days before	497	6.7
	3 days after	539	7.3

Conditions which are imperative in order to obtain successful results in coal-dust firing are:

1. The coal, before pulverizing, must be well dried, down to 1 per cent. or less of moisture. This makes it pulverize better and burn more freely. Nothing is lost in drying it separately, as all the moisture must be evaporated before coal (or any other fuel) will burn, and higher efficiency is obtained when the moisture is driven off before using. The furnace itself is the most expensive place in which to dry the coal, as the effectiveness of the whole fire is lowered by the presence of moisture.

2. The coal must be finely pulverized. The increased surface has a direct bearing upon the efficiency obtained in coal-dust firing. It is well to recognize what this increase is. C. D. Demond, Chief Testing Engi-

neer at the Washoe Reduction Works, says: "The approximate diameter of a 1-lb. lump of bituminous coal is 3 in., and its total surface one-quarter of a square foot, while 1 lb. of coal ground so that 95 per cent. will pass through a 100-mesh sieve and 82 per cent. through a 200-mesh sieve has a total surface of 8,000 sq. ft., more or less, depending upon the physical characteristics of the coal, or 32,000 times the area of the single lump."

These figures show that with finer grinding an enormous increase in surface is obtained, and, consequently, an increase in thermal efficiency. At Anaconda the grinding is done so that from 93 to 97 per cent. will pass through 100 mesh and 79 to 82 per cent. through 200 mesh. Coals with higher specific gravity will grind finer in a Raymond pulverizer. N. L. Warford, in charge of our coal-dust equipment, says: "In cement work no better work is obtained when coal is pulverized finer than 95 per cent. through 100 mesh." (This gives from 75 to 85 per cent. through 200 mesh, the percentage depending upon the physical character of the coal.) Coal thus pulverized will contain a high percentage of fine dust practically unmeasurable. As we can burn all the coal thus prepared, there seems to be no good reason for pushing pulverization beyond this point. Coal can be cheaply brought to this condition and the mills able to do this work have large capacity. Higher percentages may be obtained by the sacrifice of capacity, and consequently economy. This standard of approximately 85 per cent. through 200 mesh and 95 per cent. through 100 mesh is a practicable commercial standard and should be maintained.

3. The delivery of the coal and air to the furnace must be controlled so that the proper quantities of each may be secured. Undoubtedly, the proper method of firing pulverized coal is to admit with the fuel the exact quantity of air required for the results to be attained, and to maintain this relationship as long as results wished for are obtained. We know that in order to get the best heat efficiency from fuel a certain amount of air must be provided for combustion. Any air in excess of these requirements dilutes the gases and lowers the temperature; and insufficient air will burn part of the fuel to CO_2 and part of it to CO, which means incomplete combustion.

4. The coal must contain enough volatile combustible matter to give the required combustion. In cement work coal containing as low as 22 per cent. V.C.M. has been used. James Lord, of the American Iron & Steel Co., recommends 30 per cent. as a minimum.

5. The furnace must be properly proportioned, properly equipped, and in good condition.

6. Provision must be made for taking care of the ash formed.

At Anaconda the following equipment is installed. It is larger than is required for one furnace, but was installed with the idea in mind of finally equipping the entire reverberatory plant for coal-dust firing.

The coal from the storage bin is fed into a 30 by 30 in. Jeffrey single-roll coal crusher, where it is reduced to 1 in. maximum size. It is then taken by a belt conveyor to the foot of an elevator passing over a Ding magnetic pulley, which removes any pieces of iron, bolts, etc. It is then elevated and fed by gravity into a 40 ft. by 6 ft. 8 in. Ruggles-Coles drier. The drier consists of two cylinders, the one within the other. Blades of angle iron are fastened to the inner side of the outer cylinder and the outer side of the inner cylinder, so arranged that as the drier revolves the material fed into the space between the cylinders is lifted and dropped on to the inner cylinder and at the same time carried to the discharge end of the drier. The outer cylinder at the discharge end extends beyond the inner cylinder and has a revolving head riveted to it; on the inside of the head are buckets which lift the coal and deliver it out through the central casting. It takes a particle about 30 min. to pass from feed end to discharge end of the drier. At the feed end the inner cylinder is extended beyond the outer cylinder and, passing through a stationary head, is connected with the fire box. The gases are drawn from the fire box by means of a 72-in. Sturtevant fan, forward through the inner cylinder and back through the annular space between the cylinders to the stack. This exhaust fan is placed on top of the fire box and is connected to the drier by means of a 30-in. sheet-iron pipe. The fire-box is fed with lump coal. The capacity of a drier depends upon the moisture in the coal and the speed of the fan. With Diamondville coal, we dry 18 tons per hour. During the month of September, 1914, we used 30 tons of coal to dry 1,984.77 tons of coal.

From the drier the coal is elevated, conveyed by a screw conveyor, and discharged into a steel bin placed above the pulverizer, which is in a separate building from the drier. It is not well to have the pulverizer in the same building with the drier, for the reason that if an accident should occur, causing the coal to overflow, it might be drawn into the fire chamber of the drier and cause a fire, with possible injury to employees.

The Raymond five-roller mill is used. It has an average hourly capacity of 4½ tons. At the top of the main shaft is a rigidly attached spider which rotates with the shaft and to the arm of which the five rollers are pivotally suspended by trunnions carried in bearings in the roller housing. Both upper and lower bearings of the roller journal are provided with long, removable, phosphor-bronze bushings. The rollers are made of cast iron with chilled faces. The grinding is accomplished by the force of centrifugal motion throwing the rollers outward against the steel bull ring. A plow is located ahead of each roller and constantly throws a stream of material between the face of the roller and the ring die.

A fan is connected to this mill from which air is admitted underneath the grinding surface. The material is taken away by the air current as

quickly as it is reduced by the rolls and blown into a Cyclone dust collector placed 20 ft. above the pulverizer. The mill is thus kept free of fine material. The collector is of galvanized steel, cone shaped, and has a return-air pipe leading from it to the housing around the base of the mill. A surplus-air pipe from this return-air pipe relieves the back pressure and is an outlet for any surplus air that may enter with the feed. An auxiliary collector is placed to receive the dust escaping through this surplus-air pipe.

The finished product is discharged through a spout at the bottom of the dust collector, and is taken by a screw conveyor to a bin placed near to and above the furnace.

The coal from the bin is introduced into the furnace by means of an air current delivered through five "burners." The air current is produced by a No. 11 Buffalo fan at a pressure of 10 oz., and, by means of a pipe carrying a nozzle, is introduced into a 6-in. pipe leading into the end of the furnace. The coal dust, fed from the bin by a screw conveyor, drops upon this nozzle, which acts as a spreader, and is mixed with the air and taken into the furnace. A secondary air supply is obtained through the port holes through which the burners are projected into the furnace. These port holes are each 12 in. in diameter, which leaves an annular space 3 in. wide around each of the 6-in. pipes. By means of suitable dampers encircling the burners, this secondary air can be regulated. Another source of secondary air is through four openings between and above the burner ports, the size of the openings being regulated by putting in or taking out brick. The amount of coal fed is determined by the speed of the screw, which is easily regulated by a Reeves variable speed regulator. The entire grinding, conveying, and bin system, from the drier to the burners, is air tight, as far as practicable, with the result that the entire plant is extremely clean and free from dust.

The advantages of this method of firing are:

1. An increase in tonnage, which makes it possible to smelt the necessary tonnages with fewer furnaces.

2. The efficiency of this method is much greater than that of burning coal on grates. In burning coal on grates, there is a loss in transferring to the hearth the heat generated in the fire box. This does not occur in coal-dust firing, where the combustion occurs over the hearth itself. Another point is that none of the heat is lost in gratings from the fire box. In our practice of grate firing, this loss amounts to something like 10 per cent. of the fuel values. The efficiency is greater, due to a uniform temperature being maintained. There are no fluctuations in temperature due to firing cold fuel or to grate cleaning, or to the personal equation of the men firing the furnace.

The superintendent before leaving his work can adjust the feeding of

the coal, and by securing this adjustment with a padlock he can be certain that a definite amount of coal will be burned during his absence. If, for any reason, the furnace men need to shut off the burners, they can do so by releasing a clutch; when this clutch is re-engaged, the burner resumes firing at the original adjustment.

3. Higher temperatures than usual can be produced and maintained because the quantity of coal burned can be easily increased, and also, because there is less excess air to be heated. In grate firing 100 per cent. excess air is used, while in dust firing only 25 per cent. excess air is required. There being a smaller volume of gases, they will remain longer in the furnace, and, consequently, give up more heat to the charge.

4. Less draft is required because there is no bed of coal through which air is drawn. This is an advantage in that less cold air is drawn into the furnace through the side walls.

Tests have been run on Lochray coal, Bear Creek coal, and Diamondville coal. The coals analyzed as follows:

	Moisture	V.C.M.	F.C.	Ash	B.t.u. Dry	B.t.u. Wet
Lochray	8.0	29.3	41.8	20.9	10,350	9,730
Bear Creek	9.0	35.5	43.4	12.7	11,500	10,580
Diamondville	5.6	41.4	44.9	8.1	12,960	12,470

	Length of Test, Days	Tons Smelted per Day	Fuel Ratio	Content of Slag				Matte Cu.	Analysis of Ash			
				SiO_2	FeO	CaO	Cu.		SiO_2	FeO	CaO	Cu.
Lochray	12	409.4	5.38	37.8	42.1	5.0	0.41	39.5	39.8	13.4	5.6	1.7
Bear Creek	18	406.7	5.57	38.8	40.8	4.7	0.42	38.9	38.5	11.5	7.2	1.8
Diamondville	30	475.8	7.24	38.1	40.8	5.2	0.42	39.8	36.5	10.3	7.0	3.1

Screen Analysis of Pulverized Coal

	Per Cent. +100	Per Cent. −100	Per Cent. +200	Per Cent. −200
Lochray	2.3	97.7	12.1	85.6
Bear Creek	4.4	95.6	21.8	73.8
Diamondville	6.1	93.9	18.5	74.4

Comparison of Work of No. 7 and No. 8 Reverberatory Furnaces, September, 1914

No. 7 Furnace using Diamondville coal, grate fired.
No. 8 Furnace using Diamondville coal, dust fired.

Furnace	Tons Smelted per Furnace Day	Total Tons Smelted	Tons Coal	Fuel Ratio		
					Excluding Drier Coal	Including Drier Coal
No. 7	250.96	7,260.31	1,870.94	3.88	....	....
No. 8	475.75	14,272.52	1,984.77	......	7.19	7.08

The ash gave very little trouble. The flue connection between furnace and boiler is cleaned once per day, requiring the labor of two men from 4 to 6 hr. each day. We found the flue was easier to keep clean when using coal containing 22 per cent. ash than when using coal containing 9 per cent. ash; this difference was due entirely to the physical characteristics of the ash. In the first case the ash was light and fluffy, while in the second case it tended somewhat to sintering. Approximately, one-half of the ash of the coal floats on top of, or is absorbed by, the slag, and does not noticeably interfere with the work of the furnace. For the month of September we took from the flue 85.54 tons of ash and flue dust. This material assayed as follows:

Per Cent. Cu.	Ounces Ag Per Ton	Ounces Au Per Ton	Per Cent. SiO_2	Per Cent. FeO.	Per Cent. CaO.
3.1	5.3	0.019	36.5	10.3	7.0

Very little of the ash goes into the boilers. The boiler tubes are cleaned no oftener than are those connected to grate-fired furnaces.

Samples of gases were taken at about the center of furnace and analyzed as follows:

	First Sample Per Cent.	Second Sample Per Cent.	Third Sample Per Cent.
CO_2........................	16.0	15.0	15.0
O_2........................	2.0	3.5	3.5
CO........................	0.0	0.0	0.0

Temperatures are:

40 ft. from back end 2,250° to 2,430° F., average, 2,353°
Flue...............1,595° to 1,700° F., average, 1,645°

The labor employed in the pulverizing plant is more than normal on account of not using an economical unit. The drier is run 8 hr. per day and requires three men to shovel coal to crusher and one man to tend the

drier. With mechanical feeding of crusher, four times as much coal can be dried at the same labor cost. One man per shift of 8 hr. tends the pulverizer. When more pulverizers are put in one man can tend two of them.

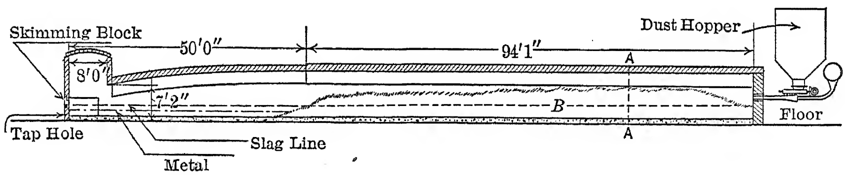

FIG. 1.—LONGITUDINAL SECTION OF COAL-DUST FIRED REVERBERATORY FURNACE.

The repairs as yet are almost *nil.* In time, as the elevators, conveyors, etc., need replacement, repairs will cost something but will never be very high per ton of coal prepared. The electric power for drier, pulverizer, fans, elevators, and conveyors totals 125 h.p. per month.

On account of not working an economical unit a fair average cost of

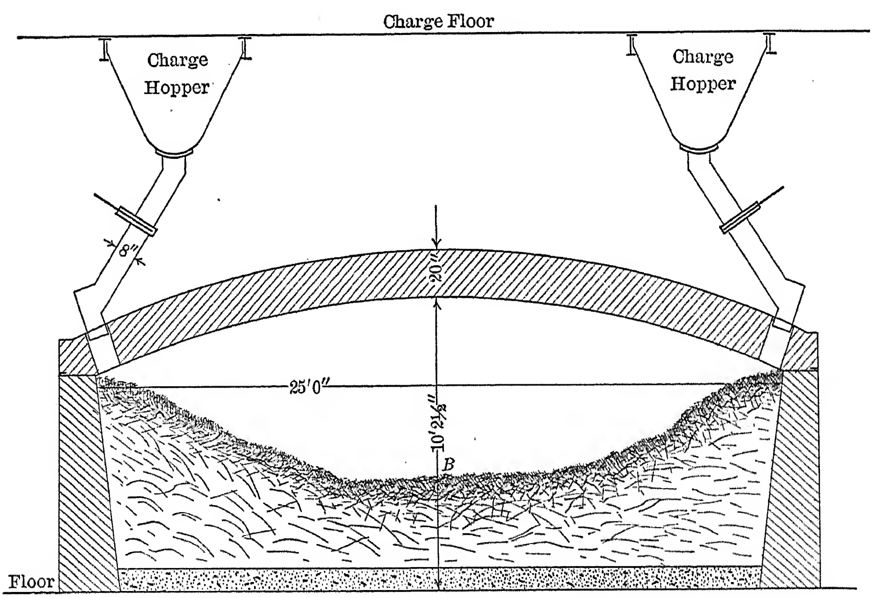

FIG. 2.—CROSS-SECTION OF FURNACE ON LINE *A–A* OF FIG. 1, SHOWING SYSTEM OF CHARGING.

preparing the coal cannot be given. The general opinion is that it can be done for 35c. to 40c. per ton of coal. This means from 5c. to 7c. per ton of charge smelted. There are three men per shift of 8 hr. doing the skimming, tapping, and charging of furnace.

The management of the works are now planning a furnace of larger

dimensions. Profiting by the work done by No. 8 reverberatory, a few changes in construction will be made. This furnace, shown in Figs. 1 and 2, will be 144 ft. long by 25 ft. wide, inside dimensions; 9 ft. 3½ in. high at the back and 6 ft. 6 in. at the front; flue area, 48 sq. ft. It will have hoppers for charging calcines and fettling material on both sides for its entire length. The tapping of matte will be from the front of furnace. The pipes leading from the charge hoppers will be enlarged to 8 in. The skimming plate will be 12 in. higher than in the other furnaces, thereby giving a larger reservoir for the accumulation of matte. The top of the skimming plate will be 24 in. in height above the tap hole. The general construction will be as before. No changes are to be made in the machinery for preparing and delivering the coal to the furnace.

The coal used at present (Oct. 16, 1914) is not as finely pulverized as heretofore. The following is an average for the past week:

Per Cent. + 100	Per Cent. + 200	Per Cent. – 200
7.5	21.1	71.4

No difference is noticed in the work of the furnaces.

The average tonnage per day for the week ending October 15, 1914, was 542.7, with a fuel ratio of 7.50.

I wish to express my thanks to all who assisted me in preparing these notes.

Coal-Dust Fired Reverberatory Furnaces of Canadian Copper Co.

BY DAVID H. BROWNE,* NEW YORK, N. Y.

(New York Meeting, February, 1915)

THE use of coal-dust fired reverberatory furnaces, or indeed of reverberatory furnaces of any description, was for the Canadian Copper Co. a matter of necessity, and not of choice. For 20 years smelting had been done in blast furnaces alone, and with the Herreshoff furnaces used prior to 1904 there was no trouble in treating fine ores. But little flue dust was produced, and this, following the time-honored custom, was wetted down and put back with the charge. Whether the flue dust really smelted, or whether it was worn out by being chased around in a circle, was a problem that troubled no one.

With the installation of modern blast furnaces and high-pressure blowing engines, in 1904, flue dust commenced to assert itself. Evidently more dust was made than could be smelted, but so many vital problems engaged our attention at that time that this minor question was pushed to one side.

In 1906, the details of blast-furnace smelting and the conversion of matte had been worked out to a satisfactory conclusion, and the ever-increasing piles of flue dust and fine ore on the stock yard demanded serious consideration. Numerous experiments in sintering, briquetting, mixing with converter slag, forming blocks of flue dust with green-ore fines and cement, and so on, were undertaken. None of these showed much promise. Our problem was still further complicated by the question of treating converter slag. The ore was basic, the slag was not needed as a furnace flux, and it was felt that under these conditions the old method of pouring slag in molds and resmelting in the blast furnace was an unnecessary expense. If the converter slag could be settled in basic-lined reverberatory furnaces, in which at the same time flue dust and green-ore fines could be smelted, two problems might thus be solved.

Reverberatory practice with our ores was, however, an unknown factor. As carried on in the West, on siliceous ores and concentrates, at least 25 per cent. of fuel was required, and even this ratio depended greatly

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upon the skill of the firemen. The lack of skilled labor; the difficulty of recovering unburned coal from the ash by water concentration in our Northern winters, and the difficulty of utilizing it, if recovered, in a plant using no steam power; the uncertainty of the effect of highly basic charges on the hearth and walls, and our entire unfamiliarity with reverberatory practice, caused us to defer our decision.

In the *Engineering and Mining Journal* of Feb. 10, 1906, S. S. Sörensen, describing certain experiments at the Highland Boy smelter, called the attention of the metallurgical world to the possibilities of pulverized coal as a reverberatory fuel. While Mr. Sörensen's experiments did not lead to the adoption of this method of using coal at the Highland Boy, they showed very clearly that increased tonnage, with decreased fuel consumption, could be attained and that such difficulties as he encountered were largely mechanical and presumably removable. Mr. Sörensen was, as far as I know, the pioneer in the use of pulverized coal in reverberatory furnaces.

His experiences were confirmed by Charles Shelby, who in an able article in the *Engineering and Mining Journal* of Mar. 14, 1908, described his experiences in the use of pulverized coal in the reverberatory furnace at Cananea. Mr. Shelby experienced much trouble by the sticking of ash in the flues and the formation of a siliceous blanket over his charge, but until blocked by these conditions he obtained better results, both in tonnage and fuel ratio, than had been obtained by grate firing. A profitable contract for the purchase of fuel oil led to the discontinuance of these experiments, but enough had been done to show that the subject was worthy of further investigation.

In October, 1909, I visited Mr. Sörensen at the Steptoe Valley smelter, of which he was then superintendent. We went over the details of the Highland Boy experiments together and agreed that with proper attention to structural and mechanical details the troubles there experienced could be avoided. In the same month I visited Mr. Shelby and discussed the difficulties encountered at Cananea. These also seemed avoidable. It was evident that if the problem could be worked to a successful issue the fuel ratio, then usually about 4 : 1, might be raised to 6½ or 7 : 1. This warranted considerable expenditure in working out the details of practice.

In visiting all the prominent Western smelters in that year, 1909, I found that the proposal to use pulverized fuel on a large scale was received with more interest than enthusiasm. As a rule, the profession was skeptical as to the expediency of starting a new plant on a practically unproved method.

During the fall of 1909, our Chief Engineer, George E. Silvester, visited the cement factories in the Eastern States in order to study the proper method of grinding and burning coal. His report confirmed our

opinion, that the process was practical, and during the winter plans were drawn for a reverberatory plant to use pulverized coal as fuel.

The mechanical difficulties encountered at Highland Boy and at Cananea consisted chiefly of two things, viz.: the stoppage of flues with accumulations of ash, and interruptions and irregularities in the coal-dust feed. It had been demonstrated in cement plants that the operations of feeding and burning pulverized coal could be made quite as continuous, as uniform, and as easily regulated as feeding fuel oil, provided only that proper methods were used in the preparation of the coal.

A plant equipped with the latest appliances for drying and pulverizing coal was therefore designed, in a fireproof building entirely separated from the reverberatory-furnace building. Especial care was taken to specify that all bins, conveyors, etc., for the pulverized coal be made as nearly dust proof as possible by the use of rubber gaskets, to eliminate the danger of dust explosions. To circumvent, if possible, the trouble from accumulations of coal ash, an entirely new arrangement of furnace flue was designed, the idea being to eliminate the several right-angled bends in common use, and provide, as far as possible, a straightaway course for the gases. In following out this idea, the skimming door was taken from its traditional position at the end of the furnace and placed on the side, entailing the sacrifice, apparently, of nothing but the tradition.

As the furnishing of steam power from the waste gases was not an essential feature of the installation, hydro-electric power being used exclusively in the plant, the waste-heat boiler was made entirely a secondary consideration, and was situated so as not to interfere in any way with the straightaway idea when not in use.

In February, 1910, Mr. Silvester and I visited the Western smelters to obtain information on reverberatory practice. Mr. Sörensen was keenly interested in Mr. Silvester's plans, in which he advised a few modifications of minor details, while approving the ideas as a whole.

In April, 1910, the Canadian Copper Co. authorized construction, and work was begun at once. As the entire site of the proposed plant had to be raised 11 ft. above the yard level, and a large amount of rock cutting and filling was necessary on the hillside where the bins and approaches were planned, active construction of the reverberatory proper was slow, and operation did not commence until Dec. 23, 1911.

As built, the original furnaces were lined with basic brick, and the hearth was an inverted arch of magnesite. The furnaces went into operation before any proper means of drying flue dust was provided, and during the winter of 1911–12 a large amount of the charge, wet and frozen as it came from the piles, was shoveled in through the doors of the furnace. All the converter slag was poured in, at first through an opening in the roof, later by means of an iron chute through a door near the fire end, as scrap is charged in an open-hearth steel furnace.

The introduction of so much cold air and cold material prevented any satisfactory fuel ratio. During the first five months 21,406 tons of cold charge and 43,463 tons of converter slag were smelted with 9,609 tons of coal. This shows a ratio of 6.7 tons of total charge per ton of coal, but only 2.2 tons of cold charge per ton of coal. However, as the cold charge was wet and often frozen, better results probably could not be expected.

The combustion of fuel was satisfactory from the start, no trouble being experienced either in grinding or burning the coal. The ash, while working on cold charges, choked and clogged the flue at the throat. This difficulty was not eliminated until later, when hot calcines were used and a larger tonnage was smelted. In general, the slower the furnace is worked the colder is the ash and the more it sticks and accumulates, while the faster it is driven the less does the ash hang back in the furnace. Under present conditions, with rapid smelting, the ash is a negligible factor.

In the summer of 1912, the roof and side walls were repaired and some facilities provided for drying the charge. In the winter of 1912, four Wedge furnaces were built to roast green-ore fines. These went into operation in March, 1913. At this date we ceased to send converter slag to the reverberatory furnaces, since with the opening of No. 3 mine the blast-furnace charge became more siliceous and slag could be used economically as a flux.

During this year very pronounced improvements were made by Mr. Agnew, then superintendent of the smelter, who with his foremen, Messrs. Kent, McAskill, and Mason, worked out and adapted to our use a modification of the Cananea system of side fettling. Long and shallow pockets were provided along the side walls, and through holes in the roof green-ore fines were fed to protect the sides. This naturally led to bricking up all the doors on the furnace, and the marked improvement which resulted from the exclusion of cold air and the insulation of the walls by a non-conducting and continuously renewed blanket of fines brought about the extension of this side-fettling system to take in almost the entire charge of calcines.

As the walls were thoroughly protected by the charge thus introduced, the use of basic brick in the walls and hearth was no longer necessary, and the next change, in October, 1913, was to the siliceous bottom and silica brick walls customary in Western smelters.

In 1914, the fuel ratio and furnace practice were steadily improving. The figures for the first three months in 1914, one reverberatory being in use, are given in the tabulation on the following page.

In the summer of 1914, a change was made in grinding the ore fines for the Wedge furnace. The ore, which was previously too coarse to make a good calcination, was treated in ball mills, and screened, so that only about 14 per cent. remained on a 20-mesh screen, instead of the former 40 per cent. This finer-crushed ore cannot be produced in sufficient

	January	February	March
Furnace days	31	28	31
Calcines, tons	10,020	9,460	10,860
Blast-furnace, flue dust, tons	906	922	847
Wedge-furnace, flue dust, tons	171	193	180
Converter slag	69	248	0
Green-ore fines and samples	1,731	1,326	2,308
Total charge	12,897	12,149	14,195
Coal	2,575	2,150	2,094
Charge per day	416	434	458
Coal per day	83	77	67
Ratio charge to fuel	5.0	5.65	6.77

quantity to keep the furnace up to its full capacity. Furthermore, when the calcines dropped, on account of this finer grinding of the ore, from 13 per cent. sulphur to 7 or 8 per cent., the production of slag increased and the production of matte fell off. These conditions, with shortage of calcines, militated against the high ratio of charge to fuel and in June, 1914, the fuel ratio was 5.35.

This historical narrative is introduced to show the gradual development of the process and the conditions which have brought about changes in the original plans.

The following notes on construction are given in explanation of the accompanying drawings, Figs. 1 and 2, which, reduced in size as they must be, are not always clear.

The area occupied by the reverberatory-furnace building was raised about 11 ft. above the level of the surrounding yard, by pouring furnace slag between concrete retaining walls, which were protected as the filling progressed by clay against the concrete. At distances of 56 ft. apart, on the center lines between furnaces, tunnels 12 ft. wide were provided in this slag foundation. These tunnels were to carry tracks so that the reverberatory furnaces built on this poured-slag area could be tapped into pots at the yard level. The furnaces are skimmed into 25-ton pots on this yard level.

Under the lines where the furnace side walls were to go, concrete footings were introduced, and between these footings transverse tie rods were laid in iron pipes and the slag pouring continued. These tie rods carry anchor plates which hold the footings under the furnace walls together and take up the lateral expansion thrust at the foot of the side buckstays. Under the furnace hearth the slag filling rises 12 in. above these concrete footings. On the concrete footings rise the silica-brick furnace walls.

The horizontal area of the furnace is 23 ft. 6 in. by 116 ft. 9 in. outside the brick work.

The side walls rising from the footings inclose 12 in. of poured slag which extends under the silica hearth. The side walls are carried up 27 in. in thickness to a height of 3 ft. 4½ in. and are continued with a thickness of 22½ in. for 5 ft. 4¾ in., making the total height of the side walls 8 ft. 9¼ in. up to the point where the cast-iron skew-back block is laid for the arch roof. This height is maintained for a distance of 34 ft. from the fire end, from which point the skew backs slope down to correspond to the slope of the arch roof referred to below.

The end or fire wall is 3 ft. 6 in. wide at the bottom for a height of 2 ft. and is then stepped back to 22½ in. at a height of 3 ft. 8 in., and again stepped back to a width of 13½ in. at a height of 6 ft. 3 in. At the other end of the furnace, commonly called the skimming end or front, the construction is very heavy to resist the end thrust of the hearth. It consists of a brick block, 6 ft. wide and 3 ft. high, which is stepped back to a width of 2 ft. 6 in. at the throat, at which point it is 4 ft. 9 in. high.

The roof at the fire end is of 20-in. silica brick. The height at the skew back is 7 ft. 9¼ in. above the bottom of the quartz hearth. The central line is 9 ft. 9¾ in. above the same point. The radius is 29 ft. 3½ in. on the under side of the arch.

When the hearth is in, the inside arch at the center is 7 ft. 9¾ in. to 7 ft. 11¾ in. above the top of the hearth and about 6 ft. 8 in. above the skim line, or 4 ft. 8 in. above the center line of the coal-dust nozzles.

This height of arch is maintained for a length of 34 ft. from the outside of the fire wall. In the next 12 ft. the arch drops 1 ft. 10¾ in., giving a height of 5 ft. 11 in. to 6 ft. 1 in. above the top of the hearth and about 4 ft. 10 in. above the skim line. This height is continued straight through to the throat of the furnace.

The 20-in. silica arch bricks are used for 34 ft. on the straight arch and for 12 ft. more on the sloping arch. The remaining portion of the roof is of 15-in. brick. As the height of this roof has been changed at various times the heights given for the roof at various points are not exactly as calculated.

There are no side doors on the furnace. As originally built, doors were set at 12-ft. centers, but these have been filled up, so that the side walls present a continuous face of silica brick 22½ in. thick.

The hearth is silica sand tamped into place. No binder was used, though better results might have been obtained had some base been introduced. After about five days' firing 50 tons of high-grade matte was put in to saturate the bottom. If steam from the silica sand came through the walls the heat was cut off for 24 hr. to allow the moisture to escape. Some patches of bottom floated up, but not enough to interfere

with subsequent operations. This bottom is almost flat, being 24 in. thick at the end walls and 22 in. thick at the tap hole, 36 ft. from the fire wall. Another tap hole is provided about 13 ft. from the fire wall.

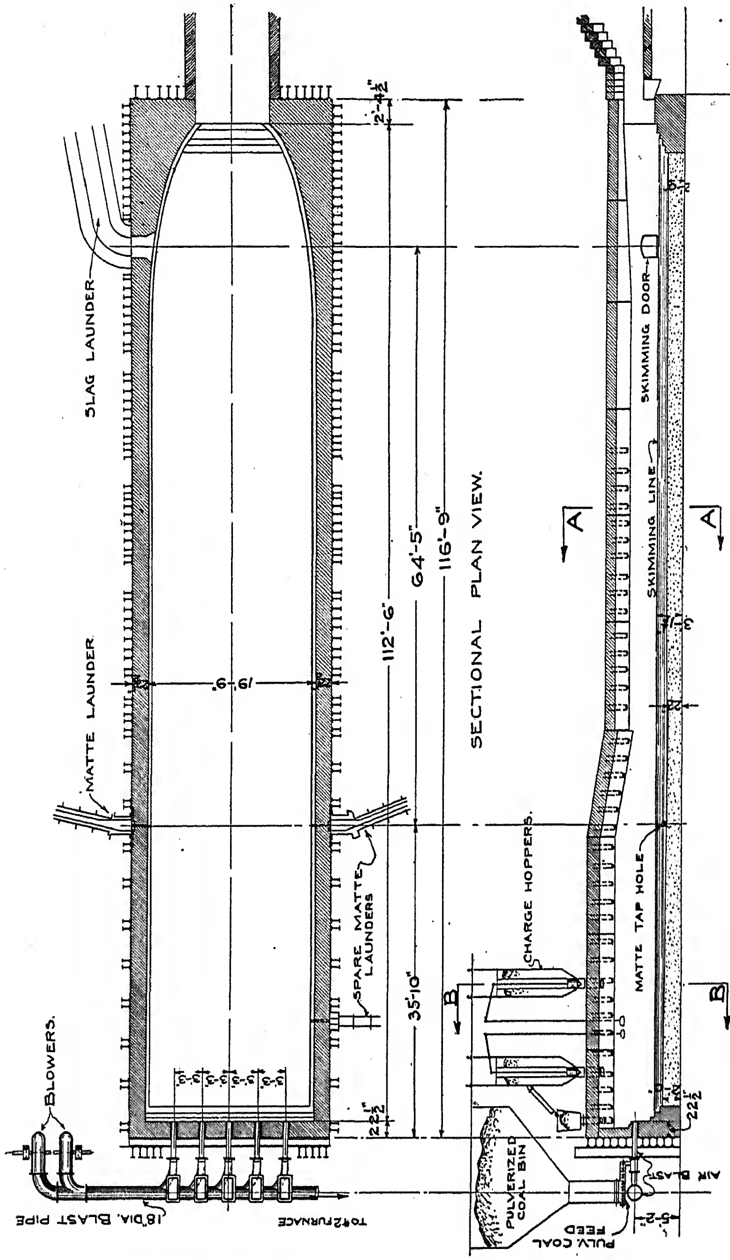

FIG. 1.—NO 1 REVERBERATORY FURNACE OF THE CANADIAN COPPER CO., COPPER CLIFF, ONT., CANADA.

In building the side walls, wood strips are introduced to provide for expansion. These wood strips, ¼ in. thick, are placed between every

four bricks on the inside and between every six bricks on the outside. As these burn out they allow the brick to expand horizontally. The arch is laid in separate sections 10 to 12 ft. wide, with the usual wooden expansion wedges 2 or 3 in. thick between the sections.

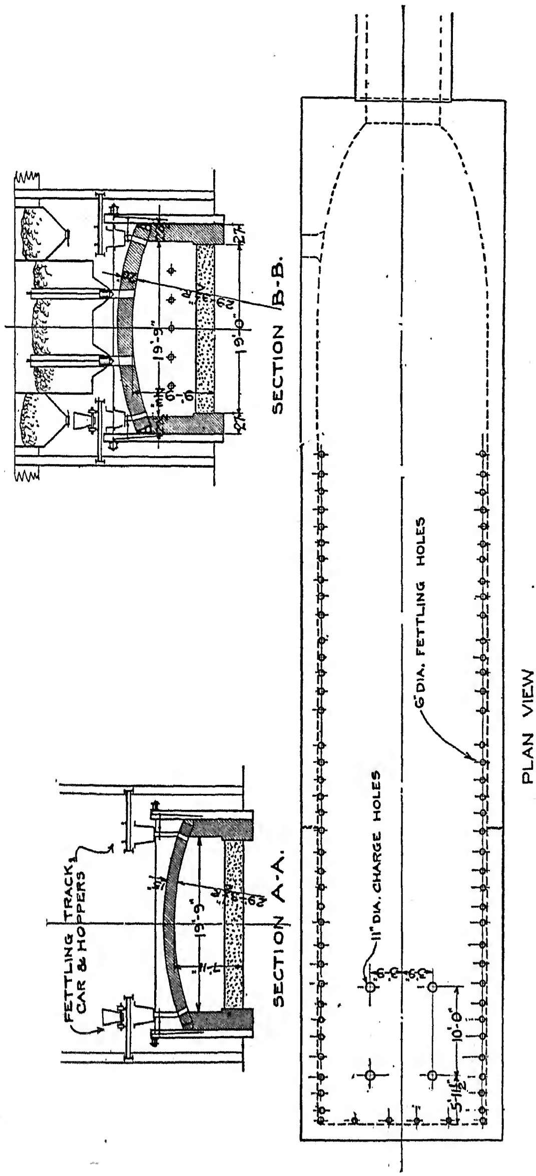

FIG. 2.—DETAILS OF THE FURNACE SHOWN IN FIG. 1.

The side walls, built as described, are carried straight to a point 26 ft. from the throat, where they curve inward, the space of 19 ft. 9 in. between

them being narrowed up along a line of gradually increasing curvature to a width of 8 ft. 8 in. at the throat. At this point the opening is 4 ft. 3 in. high at the center and 3 ft. 9 in. high at the sides. The arch here is about 4 ft. 8 in above the skimming line.

From the throat a straight flue 8 ft. 8 in. wide leads to the waste-heat boilers and to the stack. Openings are provided along the side of this flue for cleaning out any deposited ash. An opening opposite to the throat is provided by raising the bottom of this flue about 18 in. above the throat and introducing a door in the space thus formed. This is very useful for removing any accretions of ash fused in the throat. The skimming door is placed on one side of the furnace, 16 ft. 6 in. back from the throat. This door, 2 ft. 6 in. wide by 15 in. high, allows slag to run off down to a skimming line 14½ in. above the hearth at the tap hole. The slag can rise 6 in. above this line before reaching the level of the side doors now bricked up. Outside the skimming door a cast-iron clay-lined box is provided to trap any matte carried over. From this a cast-iron slag launder curves to a line almost parallel with the furnace and delivers the slag into 25-ton pots, which are brought in on a track at right angles to the furnace under the flue.

The furnace is fed in a rather peculiar way. When the furnace was started almost all the charge was introduced through two charge hoppers near the fire end, as in the usual Western practice. The first hopper delivered through two openings 11 in. in diameter, 7 ft. 6 in. apart and 8 ft. from the outside of the fire wall. The second hopper delivered through two similar openings 18 ft. from the fire wall.

At present almost all the charge is introduced through hoppers along the side walls. Directly over the side walls, at the fire end of the furnace, large bins are provided, which discharge into small bottom-dump cars. These cars run on 24-in. tracks which are supported from overhead. Under these tracks a long trough extends down each side of the furnace just above the side walls. These troughs are filled from the cars on the track above them. Each trough has openings in the bottom, 2 ft. apart, which openings communicate by a slide gate with 6-in. iron pipes. These pipes pass into holes drilled in the roof bricks which allow the charge introduced through these openings to slide down on the side walls, over which this charge forms an almost continuous blanket. As there are no doors on the furnace, and as the 6-in. pipes are clayed into the openings in the roof, it follows that no air is introduced into the furnace except what is purposely introduced at the fire end.

These pipes form a continuous line of charging holes, which extend the entire length of the furnace. The charge on the side opposite the slag door is fed all the way to the throat. On the slag side it is fed along as far as the slag door and no farther, as the cold air coming in while skimming cools the wall from the skim door to the throat and obviates the necessity

of charging beyond this point. Six similar openings are used on the fire wall.

The walls are held in place by 12-in. I-beams in pairs, with a space of 5 ft. between each pair, which form the side braces. These are wedged in at the bottom by wooden wedges against an iron strap in the concrete footings. The concrete footings are tied together as previously described by rods passing under the furnace. At the upper end the 12-in. I-beams are tied across the furnace by 1½ in. rods.

The coal dust is introduced through five pipes, 5 in. in diameter. One of these pipes is on the center line of the furnace; the others are in horizontal line with it at distances of 3 ft. 3 in. from center to center. These pipes are 5 ft. 2 in. above the bottom of the sand hearth, or 3 ft. 2 in. above the top of this hearth. They are about 2 ft. above the skimming line of the charge and the central pipe is about 4 ft. 8 in. below the highest point of the roof.

The coal used in firing is a good quality of slack. Analysis of one lot showed: Volatile matter, 34.70; fixed carbon, 55.40; ash, 9.45; sulphur, 1.30; moisture, 4.31 per cent.

This coal has a thermal value of about 13,500 B.t.u. per pound. It is about ¾ in. and under in size and contains about 7 per cent. moisture. It is dried in a Ruggles-Coles drier, 70 in. in diameter and 35 ft. long. One ton of coal burned on the grate dries 40 to 50 tons of slack coal to about 5 per cent. moisture, which falls to 2.4 per cent. moisture after grinding. About 10 tons of slack is dried per hour of running time. The coal is ground in Raymond impact mills. About 95 per cent. passes a 100-mesh and 80 per cent. passes a 200-mesh screen.

The pulverized coal is sucked by a fan to separators above the roof of the drier building, and slides down into a screw conveyor which delivers it into bins at the fire end of the reverberatory. The dust is fed from these bins by Sturtevant automatic-feed screw conveyors, one for each nozzle, the speed of which can be regulated. These screws carry the dust forward and drop it into the air nozzles about 2 ft. from the point where the nozzles enter the furnace. Any coal-delivery pipe can be closed off by a slide gate, and any screw conveyor can be stopped by disconnecting the bevel gears attached thereto. In this way any desired number of the five burners can be run, and at any desired speed within wide limits. The amount of air delivered to each nozzle can be varied at will or cut off entirely.

As a rule the five burners are in operation. Each delivers about 13.5 tons of coal dust a day or about 19 lb. of coal per minute to the furnace. The total coal blown in is about 67 tons a day.

The dust drops from the conveyors into the air pipes which carry it forward into the furnace. The air is supplied by a 4-ft. Sturtevant fan, running at about 1,300 to 1,400 rev. per minute. The air supplied by this

fan is insufficient for the combustion of the fuel. Openings are left in the end wall between the coal burners. These openings are stopped by loose bricks, so that the amount of air is readily controlled. The draft at the fire wall is about 0.25 in. of water and at the throat it has a maximum of about 1.2 in. The combustion is very good. One test made for 10 days (Jan. 9 to 19, 1914) showed the following averages:

Coal consumption, tons in 24 hr.	69.7
Gas temperature at throat, degrees centigrade	922
SO_2 and CO_2, per cent	12.3
Oxygen, per cent	6.5
SO_2, per cent	1.14

During this test the average charge was 409 tons in 24 hr. This shows a ratio of 5.9 parts charge to 1 part coal, but much higher ratios have been attained. The average for March, 1914, was 6.84. This coal ratio depends largely upon the composition of the charge and the analysis of the slag produced.

A criticism might be made of the low temperature of the gasses at the throat, 922° C. The usual practise in Western smelters is to carry a temperature of 1,200° to 1,300° at this point, and it might be thought that this low temperature indicated inefficient firing. The fact is that the heat of combustion is utilized in smelting ore along the side walls and consequently the escaping gases, having done more work than is usually the case, are relatively cold. The function of a reverberatory furnace is to melt ore, not to raise steam, and for this reason the more heat that is absorbed from the coal gases in the furnace, the more efficient the operation and the cooler the escaping gases.

The ash from this coal causes very little trouble in operating. A small amount settles on the slag, but as this slag is high in iron it is not an undesirable feature. A small amount also settles in the flue and a few hundred pounds may stick around the throat. The ash, where exposed to high heat, forms a very light pumice-like fragile mass. The throat is cleaned out daily by opening the door under the flue. During this cleaning the firing is maintained as usual.

The great advantage of coal-dust firing is the absence of the usual breaks in the temperature curve due to grating or cleaning the hearth, and as a consequence a greatly increased tonnage and fuel ratio. The operation of firing, being purely mechanical, comes under the immediate and direct control of the furnace foreman and responds instantly to his regulation. In addition to this, the peculiar method of feeding by almost continuous side charging obviates the breaks in the temperature curve due to charging or ordinary fettling. For these two reasons the chart of the temperature shows an almost horizontal line, rising or falling in almost exact concordance with the speed of the coal-feeding device.

The maximum bath of matte and slag is 22 in. deep. A constant bath of 8 in. of matte is carried. This matte lies 6 in. below the skimming plate, so that after skimming there are 6 in. of slag and 8 in. of matte left in the furnace, making a total minimum depth of 14 in. The skimming door is banked up 8 in. with sand, so that just before skimming the slag is 14 in. deep. As the charge along the side walls occupies a great deal of room there is never at any time more than 40 or 50 tons of slag in the furnace.

In rebuilding this reverberatory, or in designing a new plant, the hearth should be widened to provide for a larger body of matte, which experience has shown to be necessary. As this method of burning coal and of admitting the charge into the furnace bids fair to come into general use it is to be expected that many changes, both in construction and operation, will be introduced. My belief is that reverberatory smelting along these lines will become cheaper than blast-furnace smelting and that a wider range of ores can be used in such a furnace than in the old-style coal or oil-fuel furnaces.

Reverberatory Smelting Practice of Nevada Consolidated Copper Co.

BY R. E. H. POMEROY, MCGILL, NEV.

(New York Meeting, February, 1915)

THE statistical data given in this paper are taken from the actual performance of the No. 2 reverberatory furnace of the Nevada Consolidated Copper Co., Mc Gill, Nev., for a period of four months, from April to July, 1914.

The principal furnace dimensions are: Length, 132 ft. ½ in. at the skim-line level; average width, 18 ft., 10 in. at the skim-line level. The roof is 7 ft. above the skim-line level at the firing end. The average area in the uptake flue is 36 sq. ft., and the area above the skim line under the vulcatory arch is 24.3 sq. ft. Fig. 1 shows a diagrammatic plan and sectional elevation of this furnace. The furnace is equipped with two 400-b.h.p. Stirling waste-heat boilers set in parallel.

California crude petroleum is used for fuel and has a specific gravity of 16.5° B., flashing in open test at 199° F. The oil is heated to about 200° F. in a steam heater before going to the burners, requiring about 9 b.h.p. per furnace day to heat it from the line temperature to the burning temperature. Measurement is made by oil meter, checked daily by reservoir readings.

The oil is fed by gravity (34 lb. static pressure) to seven low-pressure blast burners of the Steptoe type (Fig. 2). The blast for these burners is supplied at about 40-oz. pressure by a motor-driven Connersville blower of 42 cu. ft. capacity per revolution.

The air supplied by the blower through the burners for atomizing the oil amounts to about 10 per cent. of the theoretical air necessary for complete combustion of the oil. The remainder of the necessary air enters the furnace through the burner openings in the firing wall. No checker holes are provided, and the charge holes in the roof are equipped with slide gates and "dog houses" to minimize air leakages.

The combustion of the fuel, as shown by gas analysis at the front of the furnace, is practically complete, the carbon monoxide in the gases being less than 0.5 per cent.

The draft at the firing end of the furnace is 0.18 in. of water. At the throat or in the uptake above the "verb" the draft is about 1 in.

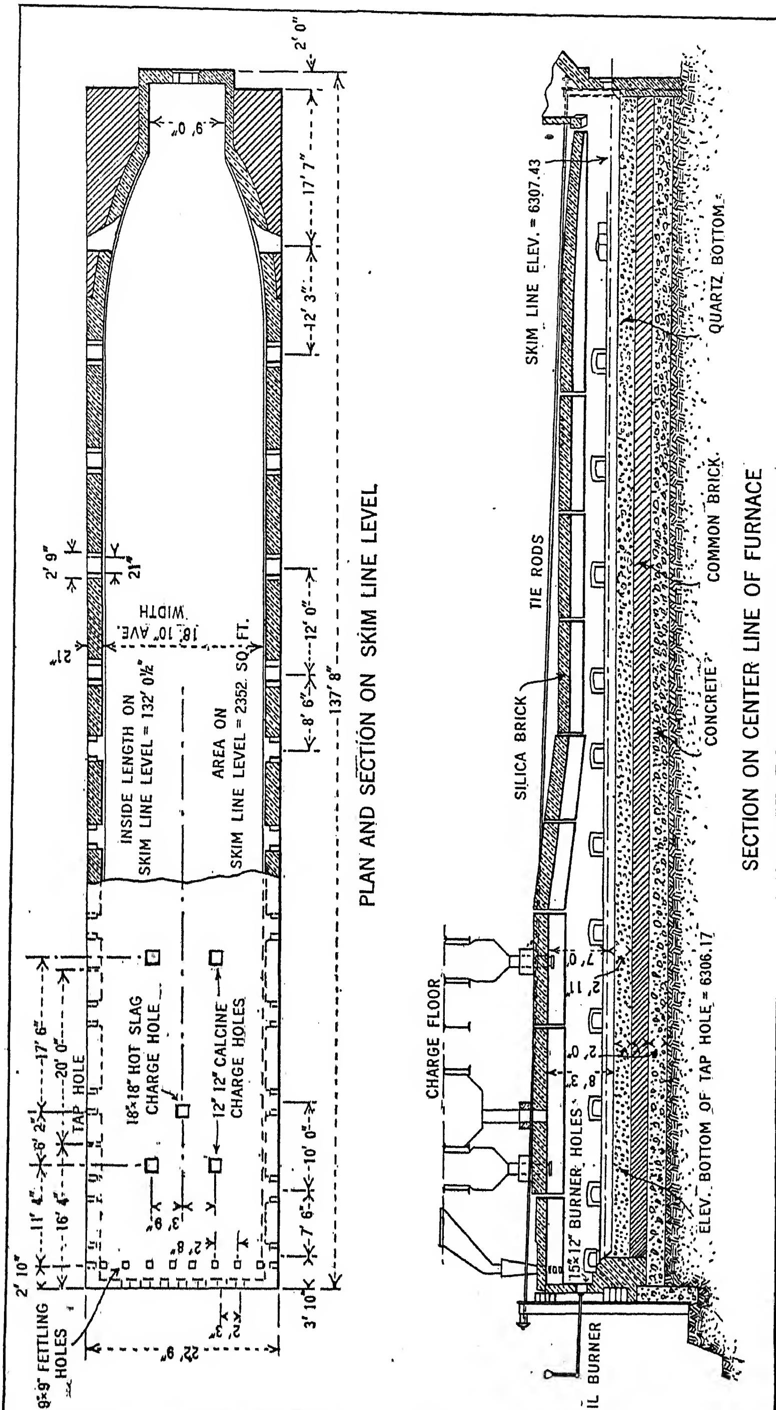

Fig. 1.—Reverberatory Furnace No. 2 at McGill.

of water, while the stack draft is 1.4 in. The top of the stack is about 320 ft. above the skim line of the furnace.

The chart shown in Fig. 3 is a gradient of flame temperatures from the firing end of the furnace to the waste-heat boilers. The temperatures at the side doors were taken with a Scimatco optical pyrometer. That at the verb was taken with a Féry radiation pyrometer, and the temperature in front of the boilers was obtained with a Brown base-metal thermo-couple.

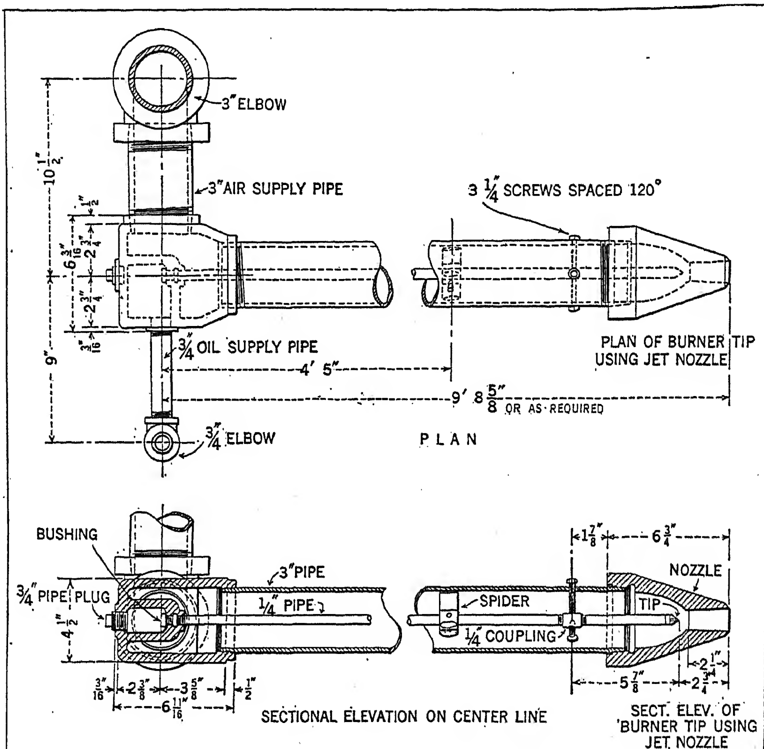

FIG. 2.—LOW-PRESSURE OIL BURNERS, STEPTOE TYPE.

From this chart the maximum temperature is found near the second side door, about 21 ft. from the burners. The only exception is during the first 10-min. interval after charging, when the combustion is delayed by the cold charge and the inert gases evolved. Here the maximum temperature is advanced about 10 ft. farther down the furnace. The effect of this delayed combustion immediately after charging is also noticeable at the front end of the furnace and in the flue to the boilers, where the temperature is highest during this first 10-min. interval.

The gradient is very uniform and the average temperature range in the furnace is not more than 300° F. between charge and charge.

The sharp drop in temperature between the last furnace door and the boilers is due to the fact that the flame is burned out at the throat of the furnace and only the gaseous products of complete combustion pass on to the boilers. Radiation from the flue and air leakage around the skimming door also contribute to this temperature drop. To minimize this loss of heat the flue has been covered by a coating, 2 in. thick, of asbestos, and the surface painted with heavy asphaltum.

The heat recovery in the waste-heat boilers is based upon the water evaporated, which is measured by water weighers, and is shown as a percentage of an empirical factor of 14 lb. of water per pound of oil. The average over this period was 34.35 per cent., or an evaporation of

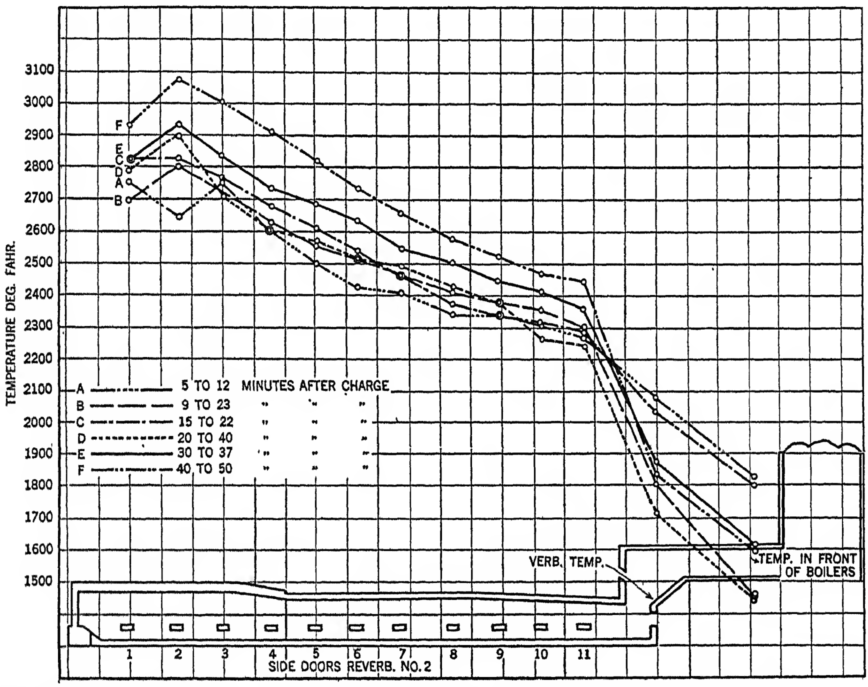

FIG. 3.—CURVES SHOWING SIDE-DOOR TEMPERATURES OF REVERBERATORY NO. 2 TAKEN AT DIFFERENT INTERVALS AFTER CHARGING.

4.81 lb. of water per pound of oil burned. The fluctuations in the steam requirements are covered by steam developed in stand-by coal-fired boilers at the power house and no effort is made to regulate the steam output from the waste-heat boilers at the expense of the oil-to-charge ratio. This permits the furnace crews to make the most economical use of the oil for smelting purposes.

Considerable fluctuation in the composition of the concentrates is unavoidable when the ore is mined by steam shovels. The ore must be treated as mined and continuous shovel cuts must be made through

the ore irrespective of its character. The conditions are, therefore, different from those encountered when treating ore from a well-developed orebody, mined by underground methods, where broken ore can be drawn from several stopes to make a desirable smelting mixture after concentration.

The percentage of copper in the concentrates varies from 4 to 10 per cent., and of sulphur from 24 to 34 per cent. The relative quantity of sulphides in the original ore is also a variable factor, hence the ratio of concentration fluctuates over a range of from 4 to 8.5 tons of ore per ton of concentrates.

As the concentrates from the mill are delivered direct to the roasters, no opportunity is offered for effective mixing, except a ground storage pile handled by locomotive crane; the concentrates must be taken to the furnaces as produced, regardless of composition, the fluxing being accomplished by adjusting the lime rock feed.

There being no blast furnace at this plant, it has been our practice for the last five years to make use of all converter secondaries for fettling the reverberatories. (See Table I for analysis.) This material is used more particularly along the firing wall or at some point where heavy material is necessary to sink into the bath and form a secure foundation at the furnace sides for siliceous fettling above and on the slag line. The principal siliceous fettling used is mill slimes or siliceous concentrates. (See Table I for analysis.) The furnaces are fettled daily by hand, using shovel and ladle, and about 30 tons of this material is used per furnace day. No brick side-door coverings are used, as each door is provided with an outside sill or shelf and the opening is piled full of fettling material.

Hot converter slag is poured direct from 10-ton slag cars on the reverberatory charge floor. The molten slag is poured on a steel grizzly and thence through a 16-in. diameter sectional cast-iron pipe into the furnace between the front and back charge hoppers. The skulls or shells from the slag pots are dumped out on the grizzly and broken down into the furnace through the same pipe with the converter slag. Thus the pots return to the converter plant empty and ready for another load.

The chemical composition of the charge components and furnace products is given in Table I.

Table II is a screen analysis of the concentrates.

The charge as fed to the reverberatory is composed of six ingredients, in proportions shown in Table IV.

The performance of the furnace is shown in Table III.

For purpose of comparison with other plants, both total and solid-charge tonnages are shown. The total charge, as the name implies, consists of all the charge, including hot converter slag. The solid charge is the total charge less the hot converter slag. The column in Table III

headed "Net oil ratio" is the "gross oil ratio" corrected for waste-heat credit, calculated as previously explained.

The oxygen ratio of the reverberatory slag, as shown from the analysis in Table I, is about 1.6 acid to 1 of base, with the alumina treated as a base. As a matter of fact, however, the slag is more acid than the analysis indicates, due to the presence of magnetic oxide of iron from the concentrates and calcines, which of course is not available as a flux for silica in a reverberatory furnace.

TABLE I.—*Analyses*

Material	Per Cent. Cu	Per Cent. SiO_2	Per Cent. Fe	Per Cent. CaO	Per Cent. Al_2O_3	Per Cent. S
Concentrates roasted.....	6.48	28.4	27.0	....	5.2	28.4
Cold secondaries.........	9.37	19.9	43.4	....	3.4	8.4
Hot converter slag.......	2.01	27.2	46.2	....	4.5	1.2
Fettling slimes	3.32	65.8	7.3	....	8.3	7.6
Lime rock..............	....	2.1	0.8	51.1	0.8	....
Roaster flue dust........	5.85	40.4	15.2	0.7	6.6	9.2
Reverberatory matte.....	27.46	0.4	40.8	....	0.5	27.2
Reverberatory slag......	0.298	42.2	28.4	9.5	6.8	0.4

TABLE II.—*Screen Analysis of Concentrates Roasted*

Mesh	+30	+40	+60	+80	+100	+150	+200	−200	Average
Weight, per cent..........	9.7	7.4	13.4	10.4	11.9	13.5	9.6	24.1	....
Copper, per cent..........	3.23	4.33	6.16	7.44	7.28	7.60	7.27	6.88	6.48

TABLE III.—*Performance of Furnace*

Month, 1914	Furnace Days	Total Charge per Furnace Day—Tons	Daily Tons per 100 Sq. Ft. Hearth Area Total Charge.	Solid Charge per Furnace Day—Tons	Daily Tons per 100 Sq. Ft. Hearth Area Solid Charge	Oil Fired per Furnace Day—Barrels	Gross Oil Ratio Bbl. per Ton: Total Charge	Gross Oil Ratio Bbl. per Ton: Solid Charge	14-lb. Basis Waste-Heat Recovery, Per Cent.	Net Oil Ratio Bbl. per Ton: Total Charge	Net Oil Ratio Bbl. per Ton: Solid Charge
April.....	30.0	694	29.53	609	25.91	371	0.53	0.61	32.48	0.358	0.412
May.....	30.8	665	28.29	581	24.72	384	0.58	0.66	34.99	0.377	0.429
June.....	30.0	659	28.03	582	24.76	372	0.56	0.64	36.43	0.356	0.407
July.....	31.0	721	30.68	628	26.72	415	0.58	0.66	33.51	0.386	0.439
Average..	30.4	685	29.15	600	25.53	385	0.562	0.642	34.35	0.370	0.421

NOTE.—The above period does not cover a complete campaign.

TABLE IV.—*Percentage Analysis of Charge Smelted*

Month, 1914	Calcines	Cold Secondaries	Hot Converter Slag	Fettling	Lime Rock	Roaster Flue Dust
April........	59.3	8.0	12.2	4.6	12.9	3.0
May........	58.7	8.0	12.6	4.8	12.3	3.6
June........	63.4	6.6	11.7	5.4	12.1	0.8
July........	58.4	7.7	12.9	5.3	13.0	2.7
Average.....	59.9	7.6	12.4	4.9	12.7	2.5

The matte fall, to borrow the term from blast-furnace practice, is about 22 per cent. of the total matte and slag produced.

The slag is skimmed off three times during each 8-hr. shift to keep the matte bath as close to the flame as possible and thus facilitate the absorption of heat by the matte from the flame, between charges.

The normal length of a furnace campaign is from 170 to 200 furnace days, and the tonnage smelted during this interval is upward of 110,000 tons. The extent of the repairs to the furnace between campaigns is about as follows:

Renewal of from 20 to 30 ft. of roof measured along the furnace. This roof is 20 in. thick and is located over the hottest portion of the hearth, around the charge holes and near the firing end of the furnace. The side walls are patched where necessary and a little brickwork is usually required at the front end of the furnace. When occasion for haste arises, campaign repairs can be completed in from four to six days between the time of shut down and the time of smelting again.

The original bottom, as shown in Fig. 1, has been worn away until the average depth of the bath below the skim line is approximately 3 ft. As the tap-hole elevation has not been changed, this gives a deep bath of matte below that level, which acts as an accumulator and regulator of heat, much as the flywheel of a steam engine acts to store up and regulate the energy developed in the steam cylinders.

The matte bath is always kept as near the skimming-plate level as possible, the smelting being done on a bath of molten matte about 3 ft. deep. This deep matte bath is of great advantage, as follows:

It serves to equalize and distribute the heat furnished by the flame, thus preventing the overheating of any portion of the hearth, with consequent boiling and damage to the silica bottom.

It becomes superheated, and when a fresh charge is dropped into the furnace it has a tendency to smelt the charge on the bottom while the flame acts on the top, thus increasing the smelting area of the furnace by attacking the charge layer on both top and bottom.

With a deep bath under the charge holes no trouble is encountered

from the charge sticking to the bottom and piling up at one point on the hearth. The charge of from 20 to 25 tons of material is dropped through four holes and spreads out uniformly over the entire hearth, utilizing all of the area under the flame, oftentimes extending down as far as the verb of the furnace.

As the furnace is fettled with great care, no break-outs or runaways of matte, due to working with this deep bath, have yet occurred.

To improve the smelting conditions the lime rock is fed to the MacDougall roasters by an outside feeder, which drops the unscreened crusher product (maximum diameter 2½ in.) on the hearth next above the bottom. It is thus thoroughly mixed with the calcines without cooling the furnace on the upper hearths where the heat is necessary for roasting, and yet is sufficiently dried and heated to prevent excessive dusting in handling.

On entering the furnace the lime rock, mixed with the other charge, comes in contact with the superheated matte bath and is rapidly decomposed with the evolution of CO_2. The charge spreads quickly over the entire hearth with violent ebullition, which thoroughly stirs and mixes it, effecting the necessary contact of acid particle with basic particle and rapidly absorbing heat from the matte bath below and from the flame above. The coarse lime rock prolongs this boiling action and is more beneficial than the fine-crushed flux.

This uniform and rapid action during the formation of the silicates gives them, when formed, ample time to liquefy and superheat, liberating the particles of newly formed matte and leaving the bath in a hot fluid condition for the next charge. The formation of charge floaters or blankets, due to slow smelting of poorly mixed charge resulting in the smelting out of the more fusible portion of the charge, is thus entirely eliminated, and only an occasional floater of siliceous material is formed from the fettling material which sometimes slips down from the furnace sides and floats away on the bath.

To recapitulate, then, I would say that the vital points in reverberatory smelting, as practiced at Steptoe, are:

Factors affecting efficient combustion of the fuel:

1. The use of a burner to give a proper atomization of the oil so as to obtain a long uniform flame without overheating any portion of the furnace.
2. Regulation of the draft so as to furnish the proper mixture of gases for complete combustion in the furnace.

Factors affecting the operation of the furnace proper:

1. Careful preparation of the charge by adequate mixing of all ingredients before charging.
2. Addition of enough lime rock, preferably coarse, to produce an active boiling in the furnace.

3. Maintaining a deep bath of molten matte to equalize and distribute the heat over the whole of the hearth.
4. Frequent skimming so as to carry only a thin layer of slag over the matte bath.
5. Operating the furnace for the best smelting conditions, ignoring the waste-heat boilers as factors in the power supply.

Factors affecting the life of the furnace:

1. The furnace roof set high over the hottest portion of the hearth.
2. Frequent fettling to protect the side walls.
3. Frequent charging and active charge mixtures to avoid floater and blanket formation requiring excessive firing.

Coal-Dust Fired Reverberatory Furnaces

Discussion of the papers of

LOUIS V. BENDER, p. 743; DAVID H. BROWNE, p. 752; and R. E. H. POMEROY, p. 764.

E. P. MATHEWSON, Anaconda, Mont.—After hearing about the success of D. H. Browne with his furnaces, we in Anaconda decided we might venture into the field of pulverized coal for reverberatory smelting. We sent a delegation of investigators to see Mr. Browne and his work, and they returned, filled with enthusiasm, and said they would recommend it strongly. We thereupon asked for an appropriation, and after some delay obtained the same and fitted up one furnace to use pulverized coal.

We copied Mr. Browne's practice and his selection of machinery for drying and pulverizing coal; and we also copied the burner he used, which was a standard article on the market. We got an expert coal-dust machinery man to design our plant, and were very glad we did so as we avoided some of the unpleasantness that had been experienced at other plants through fires and minor explosions.

I want to say, before describing our apparatus a little in detail, that we were quite familiar with the work of Mr. Sörensen, the pioneer in the use of pulverized coal for copper smelting. Our attention had been called to it by Mr. Klepetko early in the game. And afterward we were familiar with the work done at Cananea by Mr. Shelby. We sent an expert to investigate the question 10 years ago. He examined the work of Mr. Sörensen; and then went to the cement men and examined their work, and reported at great length that coal dust was the ideal fuel for them, as they took the ash from the coal and sold it as cement. He reported that coal-dust firing could not be made economical in reverberatory copper smelting, and his reasons were that a blanket of non-conducting material would cover the charge in a short time and insufficient heat would reach the charge proper, and we would have floaters over all our furnaces. The flues would be stopped by the dust that did not settle on the charge. We accepted that until our Irish friend got busy and showed us how to do it. Then we built a furnace which we thought was a slight improvement over Mr. Browne's. We put railroad tracks over our furnaces to draw the charge to the furnace. We had

long hoppers both sides of the furnace, nearly the full length, and we had pipes leading from these to the sides of the furnace through which we dropped the charge and filled the furnaces almost to the skewbacks. At the start we tried to tap our matte on the side of the furnace. But we found that the furnace seemed to bridge on the bottom and that the matte would not pass the obstruction. Then we had to stop charging in front of the matte taphole; then it struck us that it would be better to tap the matte at the front of the furnace where the slag was drawn off.

This we did, and we put an ordinary tap plate in front, 18 in. below the regular slag outlet. Then we tried to let the slag flow continuously from the furnace, tapping the matte as it accumulated in the sump in front of the furnace.

This worked very nicely. We had once in 8 hr. to take down the front door and skim off the accumulation of unfused ash which might be on the front of the furnace. This was but a few pounds and caused us no inconvenience.

We were so pleased with it that we determined at once to build a bigger furnace. We had our engineers examine the building to see what could be done. We thought we had all we could get in the building; but we learned we could lengthen the furnace a little and put the coal-conveying apparatus outside of the present building. We extended the furnace so that the hearth was 144 ft. in length, and the furnace was widened to 25 ft., inside measurement. This widening of the furnace was not a novelty, as our friends at El Paso with oil firing had reached 24 ft. But 25 ft. was the largest we could get without reconstructing our building. This furnace was put in operation early this month. We put a slow wood fire in the furnace Feb. 5, 1915. This was continued for four days; and then we put in a little coal fire, and on Feb. 9, coaled up to a tonnage of 51 tons of coal for heating this furnace. On the 10th, the day I left there, the furnace was in splendid condition, very hot, so I instructed those in charge to begin charging the furnace with calcine. They did so, as is evident from the report.

I have a telegram dated the 15th which gives the result:

"On the 10th, 345 tons were charged, and the fuel ratio was 7.52. On the 11th, 517 tons were charged, and the fuel ratio was 7.91. On the 12th, 555 tons were charged, and the fuel ratio was 7.10. On the 13th, 605 tons were charged, the fuel ratio being 7.71. On the 14th, 613 tons were charged, and the fuel ratio was 7.1." And the superintendent remarks, "Everything O. K. for full tonnage soon."

Is any one here familiar with the work done at the Steptoe Valley plant, McGill, Nev.? They have a large furnace, the performance of which is mentioned, I believe, in Mr. Pomeroy's paper. I saw that furnace in operation a couple of years ago, and it was doing some remarkable work for an oil-fired furnace.

Performance of the furnace for seven years is given on p. 769, at the bottom of the page. Total charge per furnace, 721 tons; solid charge, that is calcine without the slag, 628 tons. The balance of the charge is hot molten slag.

So they have reached a very good tonnage at that large furnace in Steptoe. Coal-fired furnaces will put through greater tonnages and will give a better fuel ratio than the oil-fired furnaces. That has been proved conclusively at Garfield. At Garfield they are abandoning oil fuel and substituting pulverized fuel throughout the plant. The price of oil, however, has something to do with it, as the oil at McGill is a little cheaper than the oil at Garfield.

Charles W. Goodale, Butte, Mont.—Apparently, the difference between success and failure in this matter of coal-dust firing rests on the degree of pulverization of the coal.

In 1906 or 1908, at the time of the first experiments in Utah and Cananea, in reverberatory smelting with pulverized coal, I understand the coal was not sufficiently fine to make the combustion complete in the furnaces, and as a consequence more or less half-consumed material fell on the charge, and that was the blanket referred to which interfered with the full effect of the heat on the charge itself.

I would like to know from Mr. Mathewson or Mr. Browne if that is not the history of the first work, and the reason why it did not succeed at first.

David H. Browne.—I cannot give the exact figures, as I have forgotten them. But I remember that at the Cananea furnace they were using an Aero pulverizer, into which the coal was fed wet and was driven into the furnace by the pulverizer itself. The coal was pulverized, and was dried by the heat of friction, and was blown in, all by the one machine. Particles of coal 0.1 in. in diameter were not uncommon, and these particles, while they burned, gave a considerable chunk of ash.

I should say that for this work almost all the coal should pass a 200-mesh screen in order to get complete combustion; and what is of more importance, to get a fineness of ash which will be carried out of the furnace by the draft. I do not suppose 5 per cent. of the ash in coal-dust fired reverberatories would stay in the furnace if the coal was pulverized to 200 mesh. In our furnaces we take out two or three bushels of ash at the front of the furnace each 24 hr. I do not suppose we take out over 400 or 500 lb. of ash in a day, if that much.

N. M. Langdon, Mancelona, Mich.—Although I did not get to Cananea until after the experiments with coal-dust firing were over, I saw the apparatus at work previously, on boiler firing; and what Mr. Browne has said about the lack of fineness of pulverization is perfectly

correct. Particles larger than 0.1 in. in diameter were given by this combined breaking and blowing apparatus, which took the lump coal and broke it, and by a crude air-blast forced it in.

James B. Herreshoff, New York, N. Y.—The description of coal-dust firing at Anaconda, the new method of charging on the sides, instead of at the center, the life of the bricks, and the size of the units, make very interesting reading.

I feel that the new method of charging reverberatory smelting furnaces is the most important improvement made in several years. The wear and tear on the furnace by this method of charging is now confined to the roof, as the sides and hearth are fully protected by the charge.

In coal-dust firing it may develop that one of the great advantages will be that fuel which cannot be burned upon the grate can be burned after being pulverized. A great deal of coal contains a high percentage of sulphur and iron, which forms clinkers. Of course, this is objectionable in the ordinary method of firing on account of the production of blowholes in the fire, which make the percentage of CO_2 in the gas low. The high efficiency of coal-dust firing is of course due to the small amount of excess air necessary to effect complete combustion.

The long life of the bricks probably results from several causes. The bricks at Anaconda are very refractory, containing a high percentage of silica—if I remember rightly, about 96 per cent.

And then again, the method of charging results in piling a good deal of material on the sides of the furnace, which acts as a flywheel, so to speak, preventing the temperature from rising too high.

Then again, being high in the back, on the firing end, the flame in the furnace does not come into contact with the roof.

They do things in a big way out West. At Anaconda they now have the largest reverberatory furnaces in the world. The furnaces are five or six times larger than any in the steel industry, and the blast furnaces are three or four times as large; and you can say the same thing about the converters.

At Great Falls, the open-hearth gas furnaces are considerably larger than those found elsewhere. And the stack at Great Falls is much larger than any other in the world.

I think it would be interesting to some of you to know that at one of the large Eastern copper refineries a large furnace has been installed for melting up cathodes. This furnace is 40 ft. long on the hearth; 19 ft. wide, and 7 ft. high, inside dimensions. The waste-heat boiler contains 11,500 sq. ft. of heating surface; that is about 1,150 boiler horsepower. The casting machine is capable of turning out 90 tons of wire-bar per hour. This furnace has already turned out 350 tons of copper

in 24 hr., and it is expected that ultimately it will turn out 400 or 450 tons in 24 hr. A furnace 30 by 16 has turned out 330 tons in 24 hr.

E. P. Mathewson.—There are many smelting furnaces of large size here running 365 days in the year, and within 30 miles of this spot there is more copper handled than any place on earth.

In connection with the remarks on reverberatory furnaces I neglected to say there was one particular point of economy in the new method adopted for charging the furnaces, and that was the delays for fettling the furnaces were eliminated altogether, saving 10 per cent. time. The charge protects the sides.

W. McA. Johnson, New York, N. Y.—I think it would be a good thing to discuss the thermal efficiency of coal-dust firing of copper calcine.

J. W. Richards.—There are not quite enough data in the paper, and I have not had opportunity to cast up the thermal efficiencies of the operations. But if we compare the reverberatory furnace in general with shaft furnaces in general, we find the shaft furnace has small losses by radiation and from the temperature of escaping gases, but a large thermal loss by reason of the percentage of unburned CO in the gas.

Comparing the reverberatory with the blast furnace, the reverberatory is at a disadvantage because of large radiation surface and radiation losses. Also, it works at high temperatures of the issuing gases. Its great advantage is perfect combustion of the fuel with the small excess of air. That is a strong point of the reverberatory furnace and enables it to make its good showing against the blast furnace.

Comparing coal-dust firing with coal-grate firing, the difference is, that in burning coal on a grate not more than two-thirds or three-quarters of the heat generated by the coal by combustion goes into the furnace. There is a loss of 25 to 33 per cent. of the calorific power of the coal, by radiation from the sides.

By coal-dust firing that is obviated. That is the essential basis of the greater efficiency of the coal-dust firing.

The other improvement which is made, namely, that of charging from the side so as to increase the heat-absorbing surface, is a very keen recognition of the fact that the heating in the reverberatory furnace is almost entirely by radiation from the flame; and therefore, if you increase the angle which the charge subtends against the flame, you increase almost in that ratio the capability of the heat absorption. In the one case you have a flat charge on the bed of the furnace; but banking it up on the sides you increase by one-half or more the area of heat-absorbing surface, and, therefore, the heat which can be put into the cold charge.

The two kinds of firing, the grate firing and coal-dust firing, are alike in this: They both give the total calorific power of the coal, as combustion is perfect; but in the one case you lose one-quarter to one-third of the calorific value of the coal by radiation.

W. McA. Johnson.—I think if Professor Richards could give the meeting a few approximate figures, we would have some way of gauging the relative thermal efficiencies.

J. W. Richards.—Roughly, it amounts to this: If you lose one-quarter of the calorific value of the coal in a fireplace, you have only three-quarters to be utilized in the furnace. The heat loss at the flue end is a constant and would be the same for both.

Let us say the charge is heaped the same way on both furnaces, the amount of heat then absorbed would be the same in both, and in one case you would have some ratio of 75 per cent. and in the other some ratio of 100 per cent.

If in grate firing you have 15 per cent., you have 15 per cent. of 75 per cent., or about 11 per cent. In the other case you get 15 per cent. of 100, or 15 per cent. net.

David H. Browne.—I want to say that most of the credit for this process is due to the boys on the spot. The best we can do is to give the boys an idea, and they make a success of it.

Almost one-half of the gain we have made in coal-dust firing is due to the way the charge is put in. Neither Mr. Sörensen nor Mr. Shelby used that. They both used the side system for fettling only. One of our foremen, now superintendent, began to use ore for fettling instead of quartz. That was so satisfactory that we discarded the charging hoppers and substituted fettling through these fettling holes. That was brought out and worked out entirely by this smelter foreman. He deserves the credit for it.

E. P. Mathewson.—I think I can help Mr. Johnson to a solution of the problem about which he asks. I can say that in the Anaconda practice we have gradually worked things down so that the combined briquet and blast-furnace expense was about equal to the combined roasting and reverberatory expense. Then on the introduction of the coal-dust fired reverberatory furnace we cut off 50c. a ton of expense, and that is probably the economy in fuel. That 50c. will mean that instead of operating blast furnaces as well as reverberatory furnaces at Anaconda, we will put our blast furnaces to one side, and concentrate all our ore with jigs, roughing tables and an improved flotation process. All our smelting will be done in coal-dust fired reverberatory furnaces.

Then, to handle the by-products, we have in mind the building of a reverberatory furnace 175 ft. in length by 25 ft. in width, in which we will

put all the molten converter slag and flux same with siliceous concentrates from the flotation process. These will carry sufficient sulphur to clean the copper out of the slag, and silica to bring the slag up to the normal composition of reverberatory slag, which can be thrown away.

We have figured that if we leave as much as 0.6 per cent. of copper in this special reverberatory slag we will beat the blast furnaces.

Henry D. Hibbard, Plainfield, N. J.—I notice that some of the roofs are 20 in. thick. Are the bricks 20 in. long?

E. P. Mathewson.—Twenty inches long.

Frank Klepetko, New York, N. Y.—I do not think you can ascribe all the benefit of the improvement in the quantity of material smelted and in the greater ratio of tonnage of ore to coal smelted to the pulverizing of the coal. I believe it is largely due to the charging on the side and firing on the charge instead of on the slag; and if someone had widened the furnace and built it higher so as to get the charge higher on the side, the grate-fire practice mght have been improved.

J. W. Richards.—You cannot get away from the fact that at least 25 per cent. of all the calorific power of the coal is lost in the fireplace by heat transmission through the walls of the fireplace, and that the gases coming in carry only 75 per cent. of it. That is put in by the coal-dust firing, and the improvement.

Henry D. Hibbard.—But a man might not have the money to install his apparatus for pulverizing the fuel. He might have, and if he adopted the method of firing on the charge instead of on the furnace he would get different results. The doubling up of the ratio is not entirely due to the pulverizing of the fuel.

E. P. Mathewson.—That was brought out by Mr. Browne. Mr. Browne called attention to the fact that the proper way to fire was to put the fire in the center of the charge, and have the charge all around the fire; and we have got as near to it as we can with the fixed furnace.

Albert F. Schneider, Plainfield, N. J.—How large a tonnage do you expect in your new big furnace?

E. P. Mathewson.—We expect to get 750 tons through the furnaces daily, with a ratio of 7.5 to 1.

F. C. Newton, Maurer, N. J.—I was interested in what Mr. Browne said about the amount of dust or ash left in the furnace. That problem has come up in the casting of copper; and one argument against the use of coal-dust firing in casting anodes or casting wirebars was the possibility of their dropping in and being slagged by the copper oxide, and requiring more copper to be in circulation back of the blast furnace. That is a

very interesting point. If only 5 per cent. of the ash falls on the bath, that is practically negligible in the casting of copper.

David H. Browne.—I do not want to be understood as limiting this to 5 per cent. I said two or three barrows of ash a day (about 500 or 600 lb). are raked out. This ash is like pumice stone; it sticks a little at the throat of the furnace, where we have a rake-out door to clean out this accumulation.

F. C. Newton.—In the anode furnace the method is to charge it by filling it with as much copper as the charging machines can handle, and place it in the furnace. I was wondering whether that would increase the amount of ash in the furnace, retarding the gases going through.

David H. Browne.—I do not think so.

A. F. Schneider.—In changing from the blast furnace to the reverberatory, do you expect to lose more or less copper?

E. P. Mathewson.—We expect the copper loss to be the same. The blast-furnace slags run lower in copper than those from the reverberatory furnaces, by about 0.1 per cent. of the total weight of slag. But the amount of lime added to the charge counterbalances that. The lime contains no copper.

H. S. Munroe, Concrete, Colo. (communication to the Secretary*).—A few remarks on Western cement practice in this matter may be acceptable.

In two cement plants operating on the same coal, which I have had occasion to observe more or less closely, the practice varies rather widely. In each of these plants Walsenburg slack is used, which obviates the necessity of using rolls.

In the first plant considered, the coal is dumped into a receiving hopper and elevated into two cone-bottom bins, from the bottoms of which it is drawn by two screw conveyors, delivering to a common drag conveyor. The drag passes the coal over a magnet to eliminate the iron (a very important factor, as will be shown) and the coal is fed by gravity into a Ruggles-Coles drier.

At the discharge end of the drier the coal is dumped into the receiving hopper of a No. 6 Universal Williams hammer mill, having 3/32-in. grate-bar openings. The hammer mill discharges into an elevator pit, from which the coal is elevated into a bin which feeds a No. 16 Smidth tube mill, in which 3/4 to 1 1/4 in. nut punchings are used as pebbles. This mill discharges into another elevator, which in turn discharges into a screw, taking the coal to the kiln coal tanks.

Suitable dust-chamber provision is made for dust from the drier,

*Received Jan. 5, 1915.

the discharge of which leads into the elevator pit preceding the tube mill.

This mill has a capacity of 100 tons per 20-hr. day, and the resultant product has an average fineness of 96 per cent. through 100 mesh. The power consumption, including stock piling, done by overhead screw conveyor and reclaiming from stock pile done by three longitudinal screw conveyors, each 96 ft. long, and one transverse screw conveyor 40 ft. long, is about 30 kw. per ton, and the total cost over any year, including all maintenance charges, is about 35c. per ton of coal ground.

With a greater finish-grinding capacity, the hammer mill could be eliminated on slack coal and a power economy be effected.

The addition of rolls before the drier would adapt a plant of this type to mine-run or lump coal.

The other plant mentioned has a capacity of about 200 tons per day on the same material. In this the coal is dried in a rotary drier of local manufacture and, without any preliminary grinding, is fed to a 6-ft. Griffin mill and a No. 16 tube mill, both of which turn out a product of 95 per cent. average fineness through 100 mesh.

I have not had access to cost figures on this plant, but think 27c. per ton would amply cover.

On the construction of coal-grinding plants, several salient points should be borne in mind.

1. There should be *no* wood anywhere in the construction. A coal-grinding plant is on fire somewhere at least 50 per cent. of the time it is operating, especially with an ordinarily gassy coal. This fire will be present wherever coal dust is allowed to accumulate.

2. Ample ventilation should be provided, to prevent the accumulation of a dust-laden atmosphere in the top of the building, a condition inviting explosion.

3. Power lines, where possible, should be run under or around plant, but when necessary to run inside plant, should be in conduit. Only incased lights should be used. Motors should be kept outside plant where possible, and when inside, should be only induction motors. A spark from the commutator of a direct-current motor will often provide the necessary ignition to set off the whole works. All electrical controls should be outside the possible dust zone.

4. Driers should be designed to be fired outside the plant.

5. Iron should be surely eliminated from the coal where hammer mills are used; 95 per cent. of the explosions which have occurred in the plant first considered have been occasioned by iron being heated to incandescence in the hammer mill, causing a general explosion which invariably cleaned out the building of combustibles.

There is no particular danger in the operation, when every possible precaution is exercised. I know of one plant where nine men were burned

to death while cleaning down the upper part of the mill, but this dust could not have been ignited without fire. On the other hand, another plant has operated eight years without seriously burning any one, though numerous explosions have occurred. No doubt there are many plants with as good records.

W. D. Leonard, Garfield, Utah (communication to the Secretary).—The following are some of the determining factors leading up to the investigation and final adoption of coal-dust firing for copper-smelting reverberatory furnaces at the Garfield plant.

1. Great difference between the cost of fuel oil and the cost of its equivalent heat units from coal was one of the deciding factors.

The thermal heat value of the ordinary coal is about 12,200 B.t.u. per pound. The thermal heat value of California oil is about 18,400 B.t.u. per pound.

About 1½ tons of coal are required to equal in heat value 1 ton of oil, and when the price of coal is considered it will be found that the ratio in favor of the coal is about 3 to 1.

2. The belief and final confirmation was reached that powdered coal could be burned nearly as efficiently as oil; or to put it more clearly, that the heat units required to smelt our charge would be about the same whether oil or powdered coal was used.

The essential part of the equipment required for powdering coal is as follows:

(This description of the equipment has been made as brief as possible inasmuch as Mr. Bender's paper covers the equipment in detail and our equipment and that of Anaconda are identical, with the exception of the size of drier.)

Our drier is the Ruggles-Cole No. A-8, 30 by 5 ft., rated at 10 tons per hour. We have not been able to run the drier to full capacity as our furnace and storage are not equal to full capacity of the drier. We have averaged 7.5 tons per hour; water in coal before drying, 3.7 per cent.; after drying, 1.5 per cent., and requires 1.42 per cent. of coal for drying. One Raymond 5-roller pulverizing mill rated at 5 tons per hour, together with accessory fans, conveyors, and separators. We have averaged 4.1 tons per hour and so far the mill has been satisfactory. Four Sturtevant coal burners

The plant was installed for our No. 6 reverberatory and put into operation about May 1, 1914. During the first ten days of operation considerable delay was experienced, due to minor defects, but the plant has operated very satisfactorily since.

Tests were made on various Utah and Wyoming coals and it was found in general that bituminous coals were superior to the lignites, all things being considered.

As regards the use of powdered coal in the furnace, there is very little difference to be noticed from oil firing. No difficulty has been encountered from ash accumulations in flues or boilers, or blanketing of charge. The furnace arches over the combustion chamber apparently last as well as in oil firing. The fettling, however, is smelted out near the fire end and at the bridge wall rather faster than in the case of oil.

It has been thoroughly demonstrated here that for the average use of our coal, about 10 per cent. more heat units are required to smelt a ton of charge than are required in oil firing. This was arrived at by comparing the final ratios over periods of 30 days, and finally over the whole period of operation.

This difference was not constant, however. In one case, using a short-flame lignite, the ratio of heat capacities of oil and coal was 75 to 100, and in our best run on bituminous coal, 99 to 100.

Fuel ratios have shown a wide variation, owing to the use of different coals in the furnace, also conditions and character of charge smelted (note log of furnace).

Furnace capacity is largely a function of the amount of fuel burned, or that can be burned, in a given furnace. In operating our furnace, we forced the tonnage up to 460 tons per day and could have exceeded this figure if we had so desired. But our coal ratio showed a tendency to increase, and further, we were afraid the wear and tear on the brick work was not in an equal proportion to the tonnage smelted, and therefore discontinued the practice. It was considered that about 400 tons per 24 hr. was the most economical basis for our furnace and conditions.

The control of the fire or flame in a furnace using oil or powdered coal is important with respect to fuel economy. In oil firing at Garfield we aim to clear the flame at the last door, or about 30 ft. from the skimming door. At times of charging, the flame sweeps to the throat and gradually shortens until it is barely visible at this point. Draft at this point is about 0.15 in. of water. Under this practice, determinations have shown that the gases escaping at the throat average about 2.5 per cent. free oxygen, which is equivalent to about 12.1 per cent. excess air. At Garfield we consider this good practice.

In operating with powdered coal, we use a slightly longer flame, but do not permit it to reach the boilers. When clearing the flame, as in oil firing, we have obtained an average of 5 per cent. free oxygen, which corresponds to nearly 25 per cent. excess air.

Efficiency of Coal as Compared to Oil

The oil, being an ideal fuel and very uniform in composition, heat value, etc., is used below as a standard in making comparisons. In making the tests, the powdered coal was compared to oil firing during

the same period on the other furnaces, all of which were smelting the same charge. Conditions in each case were therefore as nearly as possible identical.

The ratio of the heat units in the oil and coal used per ton of charge smelted is arbitrarily taken as an expression of efficiency.

Name	Kind	Coal, B.t.u. per Ton Charge	Oil, B.t u. per Ton Charge	Efficiency, Per Cent.
Castle Gate	Slack B	4,590,000	3,956,000	86
Rock Springs	Slack L-B	4,210,000	4,011,000	95
Black Hawk	Fines B	5,134,000	4,673,000	91
Standard	Dust B	4,795,000	4,673,000	97
Bamberger	Slack L	5,498,000	4,011,000	73
Black Hawk	Dust B	4,451,000	3,772,000	84
Mohrland	Slack B	4,715,000	4,011,000	85
Independent	Slack B	3,785,000	3,772,000	99
	Averages	4,647,000	4,110,000	88

In no case has it been observed that the efficiency as considered above has exceeded that of oil.

No well-defined reasons have yet been advanced for these variations of efficiency.

In the second column, coals are marked "B" and "L" to designate whether the coal is bituminous, lignite, or intermediate.

Metallurgical results have not differed from those obtained in oil firing.

Our No. 6 reverberatory furnace is 19 by 112 ft. with arch 7 ft. 10 in. from the floor level, and is fed by two rows of hoppers 6 ft. 6 in. from the side walls, thus leaving 6 ft. of space between the hoppers in the center of the furnace.

Log of No. 6 Reverberatory Furnace, Garfield Plant. Powdered Coal Firing—July 22 to Sept. 1, 1914

Date, 1914	Charge, Tons	Coal, Tons	Coal, Per Cent. Charge	Drying Coal, Lb.	Powdered Coal			Name	Oil on Other Furnaces	Remarks
					Ash Per Cent.	H_2O Per Cent.	Through 200-mesh, Per Cent.			
7–22	396	30	20.2	1,900	7.0	1.45	71.5	Black Hawk...	0.66	247 tons by oil.
23	439	56	15.0	1,900	7.1	1.40	71.7	Black Hawk...	0.63	54 tons by oil.
24	383	60	15.7	2,500	7.3	0.90	72.5	Black Hawk...	0.59	
25	385	70	18.2	2,000	.....	......	......	Black Hawk...	0.63	Clayed.
26	343	68	19 8	2,000	7.8	1.30	73.8	Black Hawk...	0.60	
27	347	70	20.2	1,300	9.0	1 50	75.6	Black Hawk...	0.65	
28	338	70	20.7	400	9.7	1.60	74.5	Mohrland.....	0.63	
29	290	48	17.9	2,900	7.8	0.30	71.5	Mohrland.....	0.67	22 tons by oil; clayed.
30	360	69	19.2	2,300	7.6	0.60	72.4	Mohrland.....	0.69	
31	415	77	18.6	2,200	7.9	1.30	67.7	Mohrland.....	0.74	
8–1	370	70	18.9	2,200	7.8	1.15	......	Mohrland.....	0.71	
2	392	68	17.4	1,800	8.0	1.15	......	Mohrland.....	0.62	Clayed.
3	335	74	22.1	400	.....	1.00	......	Mohrland.....	0.59	
4	312	65	20.8	2,000	7.5	0.85	......	Mohrland.....	0.63	
5	414	68	16.4	2,000	7.8	0.76	......	Independent...	0.60	
6	408	62	15.2	1,600	6.5	0.93	......	Independent...	0.62	
7	397	64	16.5	1,100	7.4	1.00	75.6	Independent...	0.63	9 tons by oil; clayed.
8	415	60	14.5	1,400	7.4	1.63	67.9	Independent...	0.63	
9	417	55	13.2	1,900	8.1	1.71	70.0	Independent...	0.63	
10	379	55	14.5	1,800	7.7	1.35	71.1	Independent...	0.63	
11	401	56	13.7	1,800	7.4	1.35	71.1	Independent...	0.60	Clayed.
12	401	56	13.7	1,900	6.9	1.52	73.3	Independent...	0.60	
13	316	48	15.3	1,100	6.5	1.44	72.3	Independent...	0.63	
14	345	55	16.0	1,700	6.8	1.49	70.9	Independent...	0.64	
15	413	60	14.8	1,900	6.4	1.27	......	Independent...	0.64	Clayed patch.
16	353	60	17.0	2,200	7.0	2.34	69 3	Independent...	0.64	Clayed.
17	370	60	16.3	1,900	6 2	1.33	72.5	Independent...	0.64	
18	376	55	14.6	600	6.8	1.71	70.7	Independent...	0.63	
19	370	55	14.8	2,000	.....	1.54	......	Independent...	0.60	Bridge clayed.
20	414	56	13.5	2,200	7.8	1.15	69.7	Independent...	0.62	
21	378	56	14.8	2,000	6.2	0.94	......	Independent...	0.61	Clayed patch.
22	379	58	15.3	2,200	7.8	2.18	......	Mohrland and Independent.	0.60	
23	399	60	15.1	600	6.7	2.21	......	Black Hawk...	0.55	Clayed.
24	440	56	12.7	1,900	8.5	1.65	......	Hiawatha.....	0.62	
25	418	55	13.2	2,000	7.1	1.38	70.2	Independent...	0.62	
26	407	60	14.8	2,000	8.0	2.25	73.8	Independent...	0.61	Clayed.
27	389	62	15.9	2,000	6.0	3.80	72.8	Black Hawk...	0.67	Did not dry.
28	373	55	14.7	1,500	6.8	1.94	71.9	Black Hawk...	0.56	
29	327	55	16.9	1,600	7.2	1.90	72.0	Mohrland.....	0.70	Bridge clayed.
30	330	55	16.6	1,500	8.5	2.32	74.5	Mohrland.....	0.65	
31	Furnace shutting down for repairs.									
Avg.	378	60	15.8	1,755	7.4	1.47	71.9		0.63	

Average per cent. coal through 200 mesh, Sept. 10 to Oct. 1, 77.1.

Effect of Zn_3Ag_2 upon the Desilverization of Lead

BY F. C. NEWTON, MAURER, N. J.

(New York Meeting, February, 1915)

REFINERS of lead by the Parkes process have always been solicitous of recovering the zinc used in the desilverization, and justly so, as the loss in zinc constitutes one of the heavy costs in this method of refining. Part of this loss is due to the absorption of zinc by lead, and in the present state of knowledge is not recoverable as metallic zinc. The remainder of the zinc, which finds its way into the precious-metal-bearing crust, is recoverable and naturally engages the operators' most exacting attention.

Careful elimination of the impurities in base bullion has enabled the refiner to produce a zinc crust better adapted to retorting, which reflects favorably in the zinc recovery; and to produce a richer crust, thereby lowering the zincking cost, as the increased efficiency of the zinc results in less of it being in circulation.

It is obvious that if the ratio of concentration be increased, a lessening of the amount of crust formed will ensue, throwing less work on the retorts and diminishing the amount of retort metal sent to the cupels, thus making an actual money saving in all these operations.

In the effort to obtain a rich crust, the metallurgist has increased the heat to a considerable degree, upon the assumption that at the higher temperature the zinc-silver alloy could be squeezed free from the adhering lead.

A further possibility for improvement suggested itself in the freezing-point curve of Prof. H. C. Carpenter, upon zinc and silver. Study of this curve disclosed the existence of a definite chemical compound, Zn_3Ag_2, with a freezing point of 665° C., a promising compound for practical application.

In the experiment undertaken it was endeavored to force this compound into the crust, hoping thereby to obtain a higher concentration than formerly. It was recognized that in bringing a large kettle of lead to such a heat, more fuel would be used, the life of the kettle would perhaps be shorter, and other disadvantages would be experienced. As a commercial operation, the project would have to stand on its own feet,

so the records were carefully kept to balance the advantages against the disadvantages.

The first experiment, however, showed such refinement unnecessary, as it was conclusively indicated that the melting point of the zinc-silver compound was so modified as to nullify its feasibility as a commercial factor.

In the first experiments the bullion in the kettle was heated to 705° C., 40° above the melting point of the zinc-silver compound to be formed, and the zinc stirred in as rapidly as possible. Great care had to be taken in adding the zinc to the red-hot bullion, as in more than one instance the zinc took fire and spoiled the experiment. The work about the hot kettle was very trying to the men, and had we been successful in the endeavor to make a rich crust, it is doubtful if we could have held our kettle crew to the work. After the zinc was stirred in, the kettle was cooled, the crust removed in approximately 55° C. stages with the Howard press, and squeezed at a pressure of 90 lb. per square inch. The crusts removed at the various stages were retorted separately, a dip sample being taken from the retort metal and carefully assayed. All conclusions are based upon the assay of the rich bullion remaining in the retort after distillation, and referred to as retort metal. Reasonably accurate sampling of the rich zinc crust is practically impossible.

It will be seen in the following experiments that, in every instance, the crust removed at the highest temperature, that is, just below the freezing point of Zn_3Ag_2, furnished the lowest retort metal.

Experiments

Series A

Doré contents of bullion, 191 oz. Zinc stirred in at 705° C.

	Removed at ° C.	Retort Metal, Ounces
First press	650	560
Second press	595	1,413
Third press	540	1,537
Fourth press	485	2,090
Fifth press	430	3,290

Series B

Doré contents of bullion, 253.8 oz. Zinc stirred in at 705° C.

	Removed at ° C.	Retort Metal, Ounces
First press	595	1,117
Second press	540	2,224
Third press	485	3,059
Fourth press	430	3,944
Fifth press	400	3,418

Series C

Doré contents of bullion, 315.5 oz. Zinc stirred in at 705° C.

	Removed at ° C.	Retort Metal, Ounces
First press	625	1,864
Second press	510	4,466

These tests clearly demonstrated the impracticability of removing the crust at a very high temperature. Further experiments were made to determine what beneficial result, if any, would be obtained by the greater heat. In these experiments, two kettles of metal of the same doré contents were desilverized, one heated to 705° C., and the first press removed at 535° C.; the other treated according to the normal procedure, the zinc being stirred in at 535° C. and the pressing begun immediately.

Series D

Doré contents of bullion, 263 oz. Zinc stirred in at 705° C.

	Removed at ° C.	Retort Metal, Ounces
First press	535	4,231
Second press	485	4,627
Third press	430	4,212
Fourth press	400	3,900
Fifth press	370	4,076
Sixth press	345	2,818
	Average	4,100

Same bullion stirred at 535° C., pressed at 535° C. = 3,210 oz.

Series E

Doré contents of bullion, 315 oz.

Stirred in at 705° C.; pressed at 535° C.; retort metal, 4,165 oz.
Stirred in at 535° C.; pressed at 535° C.; retort metal, 4,465 oz.

Series F

Doré contents of bullion, 318.4 oz.

Stirred in at 680° C.; pressed at 535° C.; retort metal, 5,011 oz.
Stirred in at 535° C.; pressed at 535° C.; retort metal, 5,505 oz.

In all cases where the greater heat was used for stirring and pressing, the dross and blue powder formed, especially the former, were far in excess of current practice.

The accompanying curves (Fig. 1), to some extent, show the effect of the temperature. The bullion desilverized in the various experiments was of different doré contents. It is a well-known fact that a richer crust will result from high-grade bullion upon desilverizing than from low

grade. I have endeavored to compensate for this, in the curve, by converting the doré contents of the retort metal, in proportion to the assay

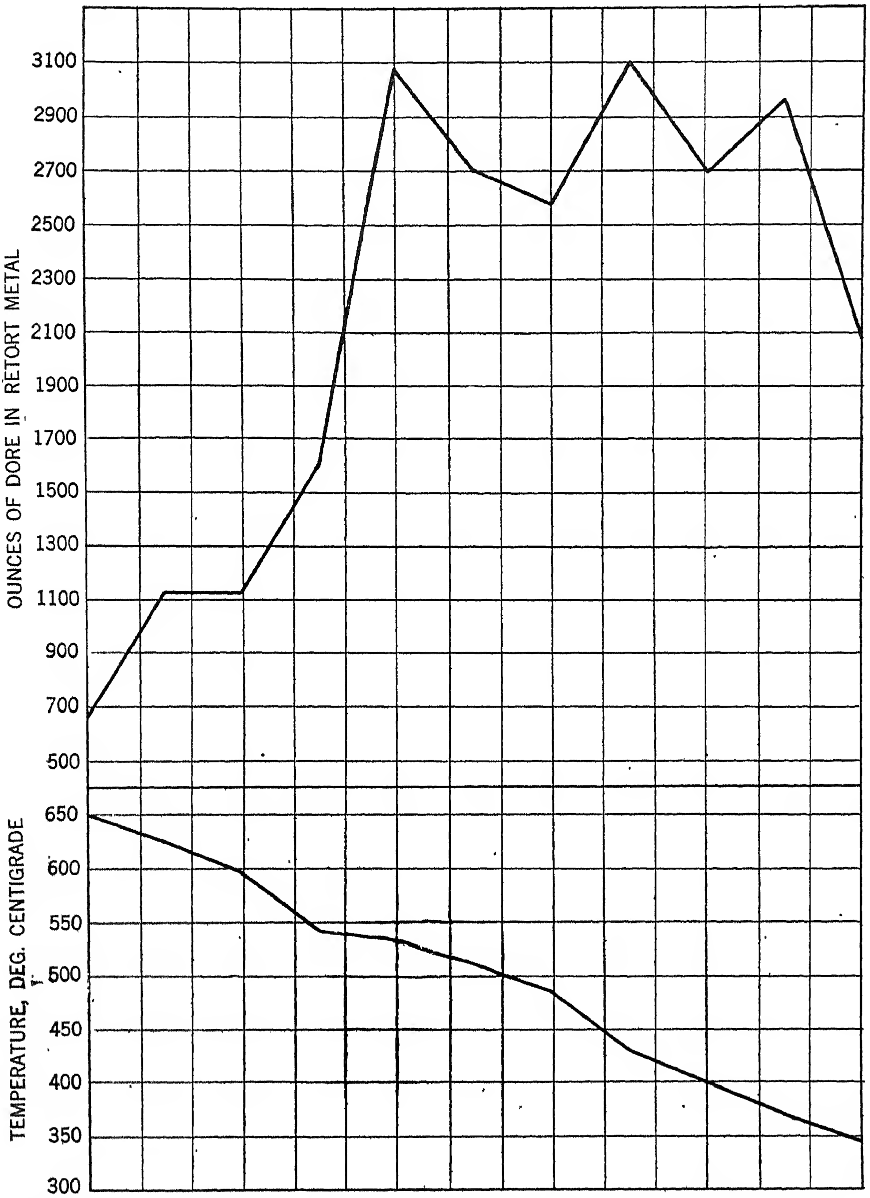

FIG. 1.—CURVES SHOWING INFLUENCE OF TEMPERATURE ON DORÉ CONTENTS OF RETORT METAL.

value of the bullion treated. A sudden drop in the grade of the retort metal at the lower temperature is due to the increase of lead contents, the lead solidifying too quickly to be squeezed from the zinc-silver compound.

The reasons for the fact that desilverization at high temperature gives a lower retort metal are not quite clear. Alder-Wright and Thompson refer to the solubility of the zinc-silver compound in lead, also its dissociation by means of lead. Kremen and Hofmeier in a paper upon Ternary Systems of Lead-Zinc and Silver-Zinc Compounds, base their investigations upon the old freezing-point curve of Petrenko. They overlooked, however, the point vital to the practical application, namely, the solubility of the zinc-silver compound in lead, or its desilverization by means of lead. A full explanation of the results obtained in the above experiments would require an extensive laboratory research upon the behavio of Zn_3Ag_2 with lead at varying temperatures. While this would be interesting, we could hardly alter the facts.

The conclusion to be drawn is that, as a commercial project, the slight increase in doré contents of the crust, which might be obtained from zincking at the greater heat, is more than counterbalanced by the attendant disadvantages. The current practice dictated more or less by the question of fuel economy and better life of kettles, has been justified as a practical and efficient procedure from the standpoint of concentration. For comfort of operation, effect upon the men, etc., all arguments are in favor of zincking at a lower temperature.

Discussion

H. O. Hofman, Boston, Mass.—The results of these large-scale experiments, which were carried on at Perth Amboy, show that there are many differences between theoretical conceptions and their applications to work on a large scale.

The author has given reasons why the results of bringing a desilverizing kettle to the high melting point of the zinc-silver alloy, Zn_3Ag_2, melting at 665° C., are not satisfactory. In all probability the alloy is dissolved in the lead; it may also be decomposed by the lead; the practical result is that this high temperature does not work. It remains now for the laboratory man to carry on systematic experiments to find out what takes place when this zinc-silver compound is brought into contact with lead at various temperatures.

I have in mind taking a number of iron tubes, mixing the alloy, Ag_2Zn_3, with lead, bringing the mixtures to different temperatures, holding them there for a given time, then chilling the tubes, opening them, and making assays and micrographs of cross-sections of the lead cylinders.

J. W. Richards, So. Bethlehem, Pa.—The curve on p. 789 would show the results more clearly if the temperatures were taken as abscissæ and the grade of the bullion carried as ordinates. In that way the

curve would not have so irregular a shape, and the results would be more easily seen.

W. McA. Johnson, New York, N. Y.—I would like to know a little more about this compound of zinc and silver.

F. C. Newton.—This compound was indicated by Professor Carpenter's work upon this zinc-silver lot of alloys, but I have never seen or made the alloy. I accepted his work and went ahead on that basis. I know very little of the properties of the compound except that it contains about 53 per cent. of silver. As a matter of fact, I have found some very interesting alloys in the retort metal, that is, the residue left in the retort after the zinc. In the retort metal, the average contents being 3,000 oz. of silver, I have taken lumps as big as or larger than a walnut of an alloy that would assay 11,000 oz. In retorting, a crust will form that runs a little high in copper if the operation is not done at the highest heat, that is, if the draft conditions are not exactly right; a hard zincky skin will form on that retort metal, which will run anywhere from 8,000 to 9,000 oz., whereas the ordinary grade of retort metal would be from 4,000 to 5,000 oz. So it is clearly indicated that these higher compounds of zinc are formed and form in the lead too. The compound is considered highly undesirable as you cannot get the zinc out of it, and the zinc losses, which ordinarily are 3 to 4 per cent. of the zinc in the crust, under these conditions would go up to 10 per cent.

H. O. Hofman.—I have not had any actual experience with this chemical compound. All I know of it is that Professor Carpenter drew the freezing-point curve of the alloys of zinc-silver and proved the existence of the compound. He did not take up the description of the compound nor its behavior when subjected to different reagents and different temperatures. I do not know that anything is known definitely about that compound.

W. McA. Johnson.—Is it soluble in lead?

H. O. Hofman.—Professor Carpenter did not touch on that question. That must be investigated. As I said a moment ago, in a year from now I hope to have some data on this question.

W. McA. Johnson.—How about the compound being analogous to yellow brass, two parts copper and one zinc?

H. O. Hofman.—The freezing-point curve of the zinc-silver alloy resembles very much that of the copper-zinc alloy. It is a curve which passes from the freezing point of silver downward to the freezing point of zinc; there is but one decided break, which shows that there exists a chemical compound. As to the behavior of the zinc-silver compound and its resemblance to that of the copper-zinc compound, I have no information.

W. McA. JOHNSON.—Is yellow brass supposed to be a definite chemical compound or an indefinite one?

H. O. HOFMAN.—As I recall the copper-zinc curve there exists a single intermetallic compound, Cu_2Zn_3, with copper 39.33 and zinc 60.67 per cent.; standard yellow brass contains copper 66.6 and zinc 33.4 per cent. The chemical compound Cu_3Zn_2 corresponds to Muntz metal, copper 60 and zinc 40 per cent.

W. McA. JOHNSON.—As a profitable research I would suggest that the boiling point as regards zinc be taken as a means of diagnosing the chemical state of these alleged zinc-silver alloys. A curve expressing such a set of determinations would give a sharp point when a change of molecular state occurred. Some years ago, I made some determinations of boiling point as regards zinc of brass, the two-thirds copper, one-third zinc alloy, commonly called yellow brass in the trade. This evolved zinc vapor at 760 mm. pressure, at 1,065° C. as against 920° C. (the boiling point of zinc). Silver resembles copper so much that it is natural to conclude that silver-zinc alloys behave analogously to copper-zinc alloys. Such a research would throw light and give clear evidence of a practical and commercial nature on such hypothetical compounds as are believed to exist. It would also have a bearing on the distillation of "zinc crusts" in the Parkes process.

J. W. RICHARDS.—The presence of the supposed maximum on the freezing-point curve is an indication of the compound. Now, that compound can be made by melting these two materials together in these proportions, and then its solubility in lead can be studied.

In studying this question, however, as to the solubility of metals in each other and their compounds, it should be borne very strongly in mind that in the melted metals those compounds do not exist. You will have the best view of the melted metal by considering it as a solution of what is present there in the definite proportions in which they exist.

There is no evidence that in the melted metal itself there is any re-solution into compounds. The melt consists only of the melted metals in mutual solution in the proportions in which they are present. The compounds are in solidified alloys, but it would be a mistake to imagine their presence in the melt. The question then comes up: What is the tendency to segregate from the melt during the operation of freezing?

EDWARD P. MATHEWSON, Anaconda, Mont.—There is one point brought out by Mr. Newton which was not brought out very strongly, and I think it has some bearing on the recent discussion. He stated there were some very rich silver alloys or compounds which segregated from the retort bullion, and he had taken out pieces as large as a walnut, extremely

rich in silver, from bullion not very rich. And he also stated that these were also found when the bullion was not very free from copper.

I think that is the secret of the segregation of these rich alloys of silver; that the copper is there always. My recollection is that these rich alloys of silver were never found unless copper was present, and the copper had something to do with this alloy.

F. C. Newton.—What Mr. Mathewson says is perfectly true. In refining lead bullion great care is taken to eliminate all the copper, as after the bullion is softened it is tapped into the desilverizing kettle, and there it may contain from 2 to 3 per cent. of copper. That bullion is brought down to the freezing point and skimmed, and in case of high copper it may be brought up and heated again and then re-chilled to get the bullion under 0.1 per cent. of copper.

The consumption of zinc is very high in case copper is present. The copper takes a certain portion of the zinc, and that zinc is not available for the extraction of silver; also, the difficulties of retorting are very apparent the instant the copper rises.

High Blast Heats in Mesaba Practice

BY WALTHER MATHESIUS, SOUTH CHICAGO, ILL.

(New York Meeting, February, 1915)

Introduction

The use of high blast heats on furnaces melting Mesaba ores is still the exception, the average blast temperatures carried on Mesaba stacks seldom reaching 1,100° F. Some 15 years ago, when the use of fine Mesaba ores in larger proportions to the total burden first came into practice, it was found difficult to employ even those blast heats which it had been possible to carry with the Old Range ores. The furnaces refused to "take" high heats, and would persistently labor with irregular stock movement, which impaired the practice and increased the coke consumption. By lowering the blast temperature, the working of the furnace became smoother, and experience taught that with the softer and more easily reduced Mesaba ores, satisfactory practice could be obtained with comparatively low blast heats. Therefore a large number of Mesaba furnaces have ever since been operated on very low blast temperatures, ranging between 800° and 1,000° F. In the face of these difficulties, furnace men were slow to recognize that furnaces could be so designed as to permit a free movement of the charges in combination with the use of higher heats. Since higher blast temperatures did not seem to offer great advantages in the smelting of Mesaba ores, investments in hot-blast stoves were not as attractive to the furnace man as labor-saving devices and means of obtaining greater tonnages.

The effect of this preference is keenly felt at the present time. It has brought about the peculiar situation, that the majority of our furnace plants (among them some of the most progressive) lack modern and efficient stove equipment. The result is that, even where the proper furnace lines have been adopted and conditions are now favorable to the successful employment of higher blast temperatures, the heats required for the greatest fuel economy are not available.

In an endeavor to obtain results, even with inadequate equipment, we find that, at many plants, the stoves are worked far beyond their capacity. This leads, in many instances, to a situation in which the coke saving in the

furnace is offset by the losses of heat carried away in the waste gases escaping from the overburdened stoves. Quite frequently, 40 per cent. and more of the total gas production is used in heating the blast, thereby seriously curtailing the amount of gas available for conversion into power. The more efficiently this power is generated, and the greater the demand for it, the higher becomes the value of the furnace gases, and the more necessary it is to economize the gas used in the stoves.

There are furnace plants which have raised their blast temperature to an average of 1,000° to 1,100° F. and are obtaining very low fuel consumption with Mesaba ores, by means of good furnace lines, suitable coke, correct distribution, and other essentials. From considerations of the high value of the surplus gas on the one hand, and lowered stove efficiency under forced operation, on the other, it may have appeared uneconomical to attempt higher blast heats, quite apart from the theory according to which the fuel saving to be expected by raising the blast temperature above a fixed range varies inversely with the reducibility of the ore burden, and rapidly decreases with increased temperatures. Accordingly, occasional attempts to carry still higher heats were, as a rule, abandoned.

But actual practice in recent years has proved a different conclusion, having shown that a considerable coke saving is still obtainable when the heats are raised above the 1,100° line. I make this assertion, notwithstanding the fact that some experiences, under conditions unfavorable to the use of high heats—conditions such as are inherent in raw materials, furnace lines, and methods of operation—may have tended to indicate the contrary.

In confirmation of this point I may mention that, during the last year, the blast temperatures at two of the Bessemer furnaces at the South Chicago plant of the Illinois Steel Co. were raised above the 1,100° line, and for many months temperatures from 1,100° to 1,300° F. were employed, with the result that the coke consumption was reduced to extraordinarily low figures without causing any operating difficulties. Data from the practice of these furnaces are shown in Table I.

Since it has been shown, therefore, that with proper furnace conditions it is not only possible, but profitable in coke economy, to carry high heats on Mesaba furnaces, the next problem is to generate these heats in the most efficient way, and without losing through inefficient stove operation the advantage thus to be gained. In other words, the problem of adapting furnace lines, raw materials, and operation to the new practice having been proved capable of solution, efficient heat generation, through suitable stove construction and operation, is of paramount interest to-day.

Stove Construction

It is not my intention to comment here on the various types of hot-blast stoves, or to discuss the advantages of any special design. I shall give

TABLE I.—*Data Concerning the Performance of No. 4 and "E" Blast Furnaces at South Works, Illinois Steel Co.*

Date	Average Daily Product	Pounds per ton: Ore, Scale, Etc.	Pounds per ton: Coke	Pounds per ton: Stone	Pounds per ton: Scrap Used Over Product.	Pounds per ton: Flue Dust Produced	Per cent. Mesaba	Cu. Ft. of Wind per Min. at Engines	Average Blast: Temp.	Average Blast: Press.	Iron Analysis: Si.	Iron Analysis: S.	Actual Yield
1914		Blast Furnace No. 4.											
*Feb....	290	4,572	2,464	1,029	—54	480	62.6	29,600	1,085	11.1	1.56	0.031	50.05
Mar...	518	4,062	1,744	781	17	193	67.4	41,140	1,170	15.7	1.29	0.025	54.76
Apr....	543	4,003	1,702	752	73	227	66.5	42,950	1,196	16.6	1.23	0.027	54.40
May...	530	4,076	1,733	785	12	152	61.4	46,260	1,161	15.4	1.24	0.028	54.68
June...	572	4,099	1,699	804	13	82	94 2	48,970	1,243	15.6	1.35	0.032	54.36
July....	560	4,196	1,765	858	—13	109	95.8	47,670	1,164	15.4	1.33	0.032	53.65
Aug....	549	4,190	1,828	884	—23	81	96.4	44,950	1,129	15.1	1.37	0.033	53.91
1913		Blast Furnace E											
Sept...	525	4,012	1,691	677	118	169	92.9	47,750	1,225	15.1	1.30	0.034	53.24
Oct....	514	4,047	1,711	774	113	171	93.0	46,390	1,251	16.2	1.48	0.043	52.85
Nov....	467	3,850	1,783	827	131	238	82.7	46,790	1,157	16.4	1.51	0.039	55.22
†Dec....	536	3,852	1,705	780	109	201	83.7	45,510	1,250	15.7	1.33	0.041	55.69
1914													
†Jan....	463	4,067	1,821	819	55	299	82.9	43,350	1,230	15.4	1.29	0.038	53.82
Feb....	489	4,216	1,850	827	11	245	74.4	42,940	1,200	16.6	1.42	0.049	52.82
Mar....	559	4,144	1,815	790	18	197	66.3	44,120	1,113	17.9	1.31	0.037	53.63
Apr....	582	4,035	1,742	775	23	234	65.9	43,950	1,146	17.7	1.27	0.028	54.92
May...	538	4,205	1,890	868	18	161	51.5	45,740	1,059	16.4	1.43	0.029	52.87
June...	558	3,993	1,894	856	22	81	78.7	46,190	1,009	15.4	1.40	0.030	55.61
July....	548	4,073	1,876	875	—22	108	95.5	46,040	1,128	15.7	1.35	0.036	55.47
Aug....	520	4,061	1,981	893	—21	79	96.2	45,680	1,098	14.8	1.45	0.036	55.60

* Blown in, Feb. 16, 1914. Tonnage produced on lining to Sept. 1, 104,350 tons. Pounds of coke per ton of product, 1,773. Average phosphorus, 0.075. Average manganese, 0.60.

† Banked, Dec. 24, 1913, to Jan. 10, 1914, inclusive. Blown in, Feb. 11, 1912. Tonnage produced on lining to Sept. 1, 448, 139. Pounds of coke per ton of product, 1,933. Average phosphorus, 0.074. Average manganese, 0.57.

only a general outline of the conditions essential for economically successful stove practice.

When at an existing furnace plant the use of higher blast temperatures is contemplated, the simplest means is often deemed to be an enlargement of the stove capacity by building an additional stove. This will, of course, increase the available heating surface and produce a higher blast temperature, but not without considerably decreasing the efficiency of the hot-blast equipment. Besides complicating operation, by increasing the number of valves and pipes, it adds to the radiating surface of the stoves, and thereby to the radiation losses; for instance, where a fifth stove has been erected, these will increase at least in direct proportion, *i.e.*, 25 per cent.

When generating higher blast heats in the above manner, the average temperature of each stove necessarily becomes higher. This means a further decrease in efficiency, since a higher stack loss is the unavoidable result. With the average stove temperature the temperature of all the radiating stove surfaces also increases, causing another considerable loss of heat, which is better realized when it is remembered that radiation increases as the fourth power of the temperature.

In spite of all these disadvantages, many blast-furnace managers have decided that the erection of a fifth stove is the best means of increasing their stove-capacity, for two obvious reasons: (1) because it can often be done with practically no other changes to existing equipment; and (2) because it insures a greater regularity of blast temperature.

For instance, where five stoves are available, it is possible periodically to cool off each unit completely for cleaning the checker work, without any loss of heat to the blast furnace. Between cleaning periods, two stoves can be operated on wind simultaneously, and three stoves on gas, which method permits a constant blast temperature to be maintained, at a higher level and with less stove changes than could be obtained by by-passing cold blast into the hot-blast main through a mixer valve. It is also claimed for this practice that each stove can be cooled, without decreasing the blast temperature on the furnace, considerably lower than is otherwise possible; and that thus the heat interchange from gas to checker work and from checker work to blast becomes more active, resulting in lower stack temperatures and better efficiency. This may be true, as long as this method is compared with ordinary practice on the same stoves, and each stove is considered individually. But it does not hold true for the set of stoves as a whole, the heat-receiving and forwarding capacity of which is really reduced to but little over that of a four-unit stove plant, while it suffers all of the radiation losses of a five-unit plant.

That the possibility of periodically cleaning each stove without heat losses to the furnace is a considerable advantage over the old

practice is not disputed. Still, the five-stove practice, like any practice with dirty gas, entails the tremendous disadvantage that each stove has to be completely cooled off when its turn for cleaning comes. The working periods between these cleaning operations can be lengthened by the well-known method of blowing the checkers off with steam; but it is impossible thus to eliminate the trouble entirely. Each time a stove is cooled off, its total heat content is lost; but the more serious objection is the contraction of the brick work throughout the stove, and its re-expansion during the following period of heating, which open the joints and cause the brick to crack, the checker work to shift, and the supporting arches to crumble. In many instances these disturbances lead to the actual breaking down of the brick work, and always hasten the final destruction of the lining.

Mechanical cleaning, too, injures the stove lining, especially the top checkers and the combustion chamber, where the flue dust carried in with the gas sinters, often melts and fuses with the brick work.

While it appears that having a fifth stove available is beneficial to the furnace practice, in allowing the regular maintenance of higher heats, yet this benefit is gained at the expense of stove efficiency, because the chief evil has not been removed, namely, the accumulation of flue dust in the stove checkers, which constitutes the greatest impediment to efficiency in stove practice. The only way to eliminate this definitely is to prevent the flue dust from entering the stoves, *i.e.*, to wash the gas thoroughly. It can be considered an established fact that it is impossible to free the gas from its dust content sufficiently by dry-cleaning systems alone, except by filtering. The latter process seems to be rapidly gaining prominence in localities where the water supply is limited. In the majority of cases the washing of stove and boiler gas in towers or rotary machines seems to be the most desirable method from an operating as well as economical point of view.

In Mesaba practice, the washing of the gas is specially important, and it is incomparably more beneficial than in furnaces melting hard ores. This is mainly for two reasons. By reason of the fineness of the ores and the fast rate of driving customary at Mesaba stacks, the amount of flue dust produced is high, and most of it is a very fine powder, which cannot be eliminated from the gas by any existing dust catcher. Hence the clogging of the checker openings proceeds very rapidly in Mesaba practice. On the other hand, the top temperature of Mesaba furnaces being comparatively low, the loss of sensible heat, which always accompanies the washing of the gas, is reduced to a minimum, and is offset many times by the elimination of the water vapor and the resulting concentration of the combustible constituents of the comparatively lean gas.

But while, in Mesaba practice, conditions without exception favor the use of washed gas, there may be considerable room for argument to

the contrary in the case of furnaces working on coarser and harder, especially magnetite, ores. The flue dust carried out by the gases from such furnaces may be small in quantity, and of such a nature as to be easily and sufficiently eliminated in the dust catcher. Furthermore, by reason of the high temperature of the rich top gases, the loss of sensible heat through washing becomes serious, and may eventually justify a decision against the installation of a wet-washing arrangement.

Notwithstanding this exception, it is the use of washed gas which has made possible the construction of stove linings with a view to stability and heat economy only, and without regard to the consideration of stove cleaning, which, if the gas be dirty, must become the prime factor in determining the checker-work construction. The checker openings, which, where unwashed gas is used, are usually 9 in. square, can, with washed gas, readily be reduced to as small a size as the draft will allow. The checker walls, which, with unwashed gas, are not considered safe when less than 3 in. in thickness, can now be reduced to the limit set by the size of checker openings as determined by the proper relation of total brick volume to surface, and by the ability of the brick manufacturer to produce a strong, substantial firebrick. In this way, the heating surface of the checker work is increased, for instance, more than 50 per cent. by reducing the checker openings from 9 to 4 in., and the thickness of the checker walls from 3 to 2.5 in. At the same time, this construction eliminates the dead portion of the brick work, in the center of the thicker checker walls, which does not participate in the heat exchange, as is proved by the fact that during the ordinary heating and blowing periods it undergoes hardly any change of temperature.

By using washed gas and changing the stove linings along the lines indicated above, any furnace plant can increase the heating capacity of its stove equipment without adding to the radiating surface. In many cases an increase which will fully meet all heat requirements can thus be obtained.

With higher stove temperatures, better insulation of the stove shell gains importance. It can be obtained by making the outer brick walls thicker, and using for them two kinds of brick: a porous kind of highly insulating quality, next to the shell; and, inside of this, a course of common firebrick (except in the combustion chamber, where a lining of highly refractory brick is required). Under no circumstances should the depth of the insulating wall be sacrificed to the desire of increasing the heating surface. To get an idea of the heat saving which can be expected from a better insulation, one has only to consider that in most stove plants the radiation losses amount to fully 20 per cent. of the total heat generated.

If sufficient heating capacity cannot be obtained within shells of a given size, the only remedy is to enlarge them, by increasing either the height or the diameter. In building new stoves a larger diameter should

be chosen; and on existing units an increase in height is generally feasible and sufficient.

Stove Operation

With proper equipment and clean gas, stove operation becomes an entirely different problem. It is no longer largely a question of crowding as much gas and air into the stove as possible. The scientific rules of combustion can now be applied, and the stoves can be heated with a minimum amount of gas by complete combustion, proper draft regulation, and an even distribution of the combustion gases and air over the entire heating surface.

Combustion of Washed Gas

In almost all cases where the change has been made from unwashed to washed gas, difficulties have been encountered, due to the slower ignition of the latter. The cold and apparently wet gas would not ignite opposite the gas and air inlets, and the flame would burn only in the upper parts of the combustion chamber and often during its downward passage through the checker openings. Stoves would look dark and cold at the bottom of the combustion chamber, and, in extreme cases, even a large excess of air would not free the stack gases entirely from carbon monoxide.

Where such difficulties have seriously interfered with the operation, and could not be overcome by adapting the stove practice to the new conditions, their cause has generally been either insufficient washing or inadequate drying of the gas.

The former trouble is manifested by too great a difference between the temperature of the water entering and the gas leaving the washer, and by an excessive deposition of mud in the gas conduits, which, after only a few weeks' run, may clog the gas outlets or burners. The gas, which, after passing through the washers is, of course, always saturated with moisture, contains at the higher temperature a correspondingly greater amount of water vapor, which, together with the active components of the gas, has to be heated up to the ignition temperature before combustion can take place. This vapor, on account of its high specific heat, considerably lowers the temperature of combustion.

Insufficient drying of the gas after washing demonstrates itself in a similar way. The gas current, though of sufficiently low temperature, contains an excessive amount of entrained moisture, which affects the ignition and combustion in a like manner.

Both these conditions cannot be remedied at the stoves and will always injure the stove efficiency by increasing the amount of inert gases passing through the stove during the heating period, decreasing the tem-

perature of combustion and raising the amount and temperature of the waste gases, *i.e.*, the stack loss. The place to eliminate these evils is at their source, the gas-washing and drying plant.

However, it cannot be denied that even with satisfactory washing and drying, it is generally more difficult to achieve quick ignition with cold washed gas than with hot gas. This difficulty, due primarily to the loss of sensible heat, becomes more serious with a decreasing percentage of combustibles in the gas. At plants running on regular low-silicon grades of iron, and with a low coke rate, such conditions may require special attention, and sometimes special devices. To remedy this evil, various arrangements have been proposed, among them the preheating of the gas or air, or both, by the waste gases, in heat interchangers. Nearly all such remedies require a complicated installation, which is undesirable and, under most conditions, unnecessary. I can hardly conceive of a blast-furnace plant working on gas so lean as to make it impossible to get complete and quick combustion simply by intimately mixing gas and air before the ignition takes place. In most cases, a properly constructed burner answering these requirements is all that is needed.

Very efficient burners of a modified Bunsen type have recently been introduced, which, without being complicated in design, give an excellent mixture of air and gas. They utilize the method of first mixing primary air and gas outside of the stove before the point of combustion is reached, and then introducing the secondary air in the proportion required for complete combustion. In this manner a high flame temperature is reached, and all of the carbon monoxide is burned with a much smaller excess of air than is possible with the ordinary type of burner. However, where the size of stoves is entirely inadequate, or relining is not possible in the immediate future, a new method of forced combustion is suggested as the proper solution.

This method, originating in Germany, and now being tried at several American plants, consists of blowing air into the gas burner with sufficient force, so as to burn the mixture of gas and air under pressure within the stove. This not only insures a quicker combustion, but also causes a much greater volume of gas and air to enter the stove than could be drawn in by the chimney draft alone. It is claimed, and experiments tend to show, that in this manner practically as much heat can be stored in the stove in one hour as can be taken out by the blast in the same length of time. If this claim can be established, it is evident that radiation losses will be greatly reduced, since only two stoves in place of the present four would be required. The cost of installing and operating a fan appears to be a very small item, when compared to the first cost, repairs, and maintenance of two hot-blast stoves.

But the installation of a good stove burner is by no means the cure for

all evils. Its proper operation is at least equally important. Weather and draft conditions, the composition and pressure of the gas, and the temperature within the stove, all of which are subject to considerable changes, are vitally influential. Constant watchfulness on the part of the furnace management and crews is required to keep the three variables, gas, air, and draft, always properly adjusted. The importance of strict supervision cannot be over-estimated, and this should be facilitated as much as possible by frequent gas analyses, accessible and easily readable gas-pressure gauges, recording stack-temperature pyrometers, and convenient means of observing the development of the flame in the combustion chamber. The aim should always be to complete the combustion in the lower part of the chamber; the shorter the flame, the higher will be its temperature, and the brighter also the appearance of the combustion chamber walls, which can readily serve as a guide for the operator. If the combustion chamber is roomy enough to permit a free development of the flame, the danger of overheating and melting the parts of the walls opposite the burner is remote.

Distribution of Gases

Proper combustion having been achieved, the next step is to distribute the gases correctly over the entire heating surface, and to bring them into the closest contact with the brick work, there to give off their heat as uniformly and completely as possible.

To facilitate proper gas distribution, many arrangements have been proposed and applied to the original designs of both side and center-combustion stoves. Most of these devices, even if correct in principle, have achieved but little, so long as unwashed gas was used. The deposition of flue dust on top of the checker work would, in the course of a few weeks, bring gas circulation to a complete standstill in a large percentage of the checker openings. Through the remainder of the openings the gas would flow with increased speed, passing only a small part of the total heating surface with little chance for proper heat exchange, and with correspondingly high stack temperature as a result. One can readily see that, under such conditions, any attempts to improve the draft distribution by more adequately spacing and dimensioning the stack valves, by the reduction of the checker openings in such parts of the stove cross-section as are favored by the draft, or by the altering of the domc-shape, would avail little—the deposition of flue dust being, in spite of all, and above all, the predominant factor influencing the draft and the gas distribution.

With the introduction of washed gas, this condition is changed completely. The checker work can now be relied upon to retain its original free area throughout a whole furnace campaign, or at least for a number

of years. The proper distribution of the gas and the correct regulation of the draft now become a problem of the utmost importance. Researches which have been carried on in recent years seem to indicate that on a large number of stoves the deficiency in this respect is serious, affecting in some cases over 50 per cent. of the total heating surface. Such conditions can now be readily and reliably investigated and adjusted. The means to be employed to this end have been indicated above and depend almost entirely on local conditions. At present a large number of furnace managers are giving this subject their special attention; and, since the goal to be attained is well defined, considerable progress in this direction may be expected in the near future.

That the heat transmission to and from the brick work is not impaired any longer by a flue-dust coating on the brick surface, is another important benefit derived from the use of washed gas. In the cooler parts of the stove, this coating appeared in the form of a spongy, porous insulating covering, and in the hotter parts, reacted with the brick work to form an iron silicate glaze of low heat conductivity. How this reaction, penetrating to the interior, deteriorates the brick, making it brittle and easily fusible, I have already observed, remarking also that nothing can be done during operation of the stove to prevent this reaction.

This danger being eliminated, the ability to withstand such chemical attacks is no longer required for a good stove brick; and the manufacturer can now concentrate his efforts exclusively upon the development of a standard quality, which will combine strength and elasticity with highest heat-storing capacity, conductivity, and infusibility.

Great progress along these lines has been made in recent years through the manufacture of machine-pressed brick, the practical service of which is the most convincing argument against the old idea that a checker work must have pores for retaining the heat, instead of the property of taking up and giving off the heat quickly. Such pressed brick will permit a considerable decrease in thickness of the checker walls, without endangering the building strength of the brick work.

Used in conjunction with smaller checker openings, it will enable us to obtain within a given stove shell the maximum heating surface with the largest brick volume and the highest ratio of active brick surface to total free space. This means the most intimate contact between the brick work and gas or air; and, together with concentrated combustion, correct gas and air distribution and draft regulation, will result in the lowest stack temperatures and highest and most uniform blast heats; that is to say, in efficient stove operation.

This paper by no means pretends to give an exhaustive account of all possible improvements which may be applied with more or less success to stove construction and practice. Its aim is solely to awaken an interest in the subject, and invite discussion of the improvements in our

stove practice, particularly with washed gas. I believe that in no other branch of our furnace work will earnest, practical and scientific experimental work be better rewarded.

Heat Calculations

The exceedingly favorable practice which has been obtained recently on Mesaba furnaces by the use of higher heats, lends new and special interest to the theoretical side of the fuel-economy question. Research along these lines has so far not been very extensive in Mesaba practice; and it is to be hoped that the recent practical progress will give a new impulse also to scientific investigations.

Examinations of existing furnace conditions in different localities appear to be of first importance; and, as a contribution of this character, a thermal analysis has been prepared of the performance of No. 4 Blast Furnace at the South Works of the Illinois Steel Co., during the month of June, 1914. It is hoped that this attempt will invite discussion and research along similar lines, and thus do its share toward throwing some light on the question, to what extent, under modern conditions, the various reactions in the blast furnace influence the equilibrium between heat supply and demand, and through this the fuel economy.

The data of practice, and the calculations employed for the establishment of the heat balance, follow in detail.

Calculation of Operating Data

Analysis of Top Gases

	By Volume	By Weight
CO_2	14.9	22.3
CO	23.5	22.4
(1) H_2	4.1	0.3
CH_4	0.2	0.1
N_2	57.3	54.9

On the basis of above data one ton of this gas contains:

(2) $$\frac{22.3 \times 3}{100 \times 11} = 0.0608 \text{ ton of carbon as } CO_2$$

and $$\frac{22.4 \times 3}{100 \times 7} = 0.0960 \text{ ton of carbon as } CO$$

This carbon originates from:

(*a*) The carbon charged, minus the amount of carbon transferred to the iron and the amount carried off with the flue dust.

(*b*) The CO_2 content of the raw materials.
These items are calculated below:
(*a*) The total amount of coke charged during the period under consideration was

(3) 13,006 tons,

which analyzed:

	Fixed Carbon	Ash	S	P	Volatile Matter
(4)	89.00	9.40	0.59	0.008	1.35 per cent.

(5) The coke as weighed averaged 2.30 per cent. moisture, making the weight of dry coke charged:

(6) 12,707 tons.

The amount of fixed carbon charged is thus:

(7) $$\frac{12{,}707 \times 89.00}{100} = 11{,}310 \text{ tons.}$$

From this total is to be deducted:
(*a*1) The carbon carried off with the iron, which is calculated as follows:
(8) The carbon content of the metal was 4.20 per cent. and
(9) 17,146 tons of iron were produced, besides
(10) 173 tons of scrap (ladle skullings and pig-machine scrap); therefore
(11) 17,319 tons of iron (17,146 + 173) carried off

(12) $$\frac{4.20 \times 17{,}319}{100} = 727 \text{ tons of carbon.}$$

Besides the regular burden, the furnace remelted 276 tons of scrap iron, which contained:

(13) $$\frac{276 \times 4.20}{100} = 11 \text{ tons of carbon.}$$

Therefore the net amount of coke carbon carried off with the metallic product is:

(14) $$727 - 11 = 716 \text{ tons.}$$

(*a*2) During the month 628 tons of flue dust were taken out of the dust catcher of the furnace. Assuming that the gas carried over an additional 10 per cent. of this beyond the dust catcher, the total amount of flue dust produced by the furnace was,

(15) 691 tons.

(16) Since the average carbon content of the flue dust was 8.25 per cent., the amount of carbon carried from the furnace with the dust amounted to

(17) $$\frac{8.25 \times 691}{100} = 57 \text{ tons.}$$

Consequently the net amount of carbon gasified in the furnace is:

(18) $$11{,}310 - (716 + 57) = 10{,}537 \text{ tons.}$$

(*b*) The only burden constituent containing CO_2 in larger quantities was
(19) limestone, of which 6,154 tons were charged. It averaged 43.1 per cent. CO_2
(20) making the amount of CO_2 charged with the limestone $\frac{43.1 \times 6{,}154}{100} = 2{,}652$ tons
(21) which is equivalent to $\frac{2{,}652 \times 12}{44} = 723$ tons of carbon.

(22) The CO_2 content of the other burden constituents (ore, scale, and cinder) averaged about 1 per cent., giving off an additional

(23) $$\frac{31,372 \times 1}{100} = 314 \text{ tons } CO_2, \text{ which is equivalent to}$$

(24) $$\frac{314 \times 12}{44} = 86 \text{ tons of carbon.}$$

The total amount of carbon which escaped from the furnace during the month in the shape of CO_2 and CO is, therefore (see 18, 21, and 24),

(25) $$10,537 + 723 + 86 = 11,346 \text{ tons.}$$

Each ton of the top gases containing 0.1568 ton (see 2) of carbon, the total weight of the dry top gases for the month is:

(26) $$\frac{11,346}{0.1568} = 72,360 \text{ tons.}$$

The total weight of water, which was charged into the furnace with the ore, coke, and limestone, and which subsequently had to be evaporated, leaving the furnace together with the top gases, was, according to the analysis of the burden constituents,

(27) 4,665 tons.

The nitrogen content of the dry top gases (see 1) originating exclusively from the blast, and the nitrogen content of the air being known, the weight of the dry air which entered into the furnace through the tuyères can be calculated as follows:

(28) $$\frac{72,360 \times 54.9}{77} = 51,592 \text{ tons.}$$

(29) The average moisture of the atmosphere was 5.49 grains per cubic foot at 70° F., making the weight of natural air blown into the tuyères,

(30) $$51,592 + 539 = 52,131 \text{ tons.}$$

(31) The slag volume according to the daily burden calculations averaged 45.5
(32) per cent., making the total amount of slag produced, 7,811 tons.

The weight of materials passed through the furnace was as follows:

	Material	Total Tons	Per Ton of Product Including Scrap Produced	
			Tons	Pounds
	Burden	31,645	1.827	4,093
	Limestone	6,154	0.355	795
	Coke	13,006	0.751	1,682
	Product (including scrap)	17,319		
(33)	Slag	7,811	0.451	1,010
	Flue dust	691	0.040	89
	Blast (excluding moisture)	51,592	2.979	6,673
	Moisture in blast	539	0.031	69
	Top gas (excluding moisture)	72,360	4.173	9,359
	Moisture in gas	4,665	0.269	603

According to the average monthly analysis the product consisted of,

	C.	Si.	Sul.	Phos.	Mn.	Fe.
(34)	4.20	1.35	0.032	0.071	0.74	93.61 per cent.

The iron content originated from the following amounts of burden constituents:

	Material	Group 1	Group 5	Pewabic Genoa	Pit Cinder	Steel Scale	Scrap	Coke	Total
	Iron, per cent....	52.00	49.00	35.00	50.00	66.00	75.00	080	
	Stage of oxidation.	Fe_2O_3	Fe_2O_3	Fe_1O_3	Fe_3O_4	Fe_3O_4	Fe	Fe	
	Pounds per ton of product........	2,819	755	219	58	207	36	1,682	
(35)	Pounds minus 1 per cent. loss...	2,791	748	217	57	205	35	1,665	
	Pounds iron per ton of product	1,451	366	76	29	136	26	13	2,097

Consequently the iron content of one ton of product originated from the following stages of oxidation:

	Fe	Fe_3O_4	Fe_2O_3
(36)	39	165	1,893 Pounds

(37) In the same manner it results that one ton of product contained 0.74 per cent. or 16.6 lb. of manganese; 16.3 lb. of which have been reduced from MnO, while 0.3 lb. were contained in the scrap charged.

(38) The phosphorus of the product amounts to 1.6 lb. per ton and has been reduced practically all from P_2O_5.

(39) The 1.35 per cent. of silicon is equivalent to 30.2 lb. per ton, of which 29.7 lb. were reduced from SiO_2, while 0.5 lb. were contained in the scrap charged.

Heat Balance

(a) *Heat Generation*

The total weight of carbon contained as CO_2 in the top gases per ton of product is (see 2 and 33):

(40) $$0.0608 \times 9,359 = 569.0 \text{ lb.}$$

The weight of carbon per ton of product, equivalent to the amount of CO_2 originating from the limestone and burden and contained in the top gases is (see 21, 24, and 33):

(41) $$\frac{809 \times 2,240}{17,319} = 104.6 \text{ lb.}$$

Deducting item (41) from (40) gives the weight of coke carbon burned to
(42) CO_2 per ton of product: 569.0 − 104.6 = 464.4 lb.
(43) One pound of carbon through combustion to CO_2 generates 14,543 B.t.u.;
(44) the above 464.4 lb. generated, therefore, 6,754,000 B.t.u.
The amount of carbon per ton of product burned to CO is (see 2 and 33):
(45) 0.0960 × 9,359 = 898.5 lb.

(46) One pound of carbon through combustion to CO generating 4,446 B.t.u.,
(47) the above 898.5 lb. generated 3,995,000 B.t.u.
(48) The specific heat of the air blast being 0.248 and the average hot-blast tem-
(49) perature 1,243° F., the heat brought into the furnace with the blast (see 33)
(50) per ton of product is: $1,243 \times 0.248 \times 6,673 = 2,057,000$ B.t.u.

In the same manner the amount of heat brought into the furnace per ton of product with the moisture (specific heat, 0.49) content of the blast (see 33) is:

(51) $$1,243 \times 0.49 \times 69 = 42,000 \text{ B.t.u.}$$

(b) Heat Consumption

The following data were used in these calculations:

		B.t.u.
	Heat required to reduce Fe_2O_3 to 1 lb. of Fe	3,240
	Heat required to reduce Fe_3O_4 to 1 lb. of Fe	2,970
(52)	Heat required to reduce MnO to 1 lb. of Mn	2,970
	Heat required to reduce SiO_2 to 1 lb. of Si	14,090
	Heat required to reduce P_2O_5 to 1 lb. of P	10,620

The reduction of one ton of product therefore requires the following heat (see 34, 36, 37, and 38):

		B.t.u.
	1,893 lb. of Fe from Fe_2O_3	6,133,300
	165 lb. of Fe from Fe_3O_4	490,000
(53)	16.3 lb. of Mn from MnO	48,400
	1.6 lb. of P from P_2O_5	17,000
	29.7 lb. of Si from SiO_2	418,500
	Total	7,107,200

The weight of CO_2 produced by the calcination of carbonates per ton of product is, according to (20), (23) and (33):

(54) $$\frac{(2,652 + 314) \times 2,240}{17,319} = 384 \text{ lb.}$$

practically all of which was contained in the burden in the form of $CaCO_3$.
(55) The heat necessary to decompose $CaCO_3$ into CaO and CO_2 is 1,830 B.t.u. per pound of CO_2. The driving off of above 384 lb. required

(56) $$1,830 \times 384 = 702,000 \text{ B.t.u.}$$

The amount of heat carried from the furnace by the slag and iron was not determined. The figures used represent the average results of former tests under similar conditions. According to these the heat carried off by 1 lb of
(57) iron amounted to 510 B.t.u. and per pound of slag to 900 B.t.u.

The heat carried off per ton of product with the iron amounts to

(58) $$510 \times 2,240 = 1,142,500 \text{ B.t.u.}$$

and with the slag

(59) $$900 \times 1,010 = 909,000 \text{ B.t.u.}$$

The dissociation of the moisture as carried into the furnace with the blast
(60) requires, per pound of water vapor, 5,760 B.t.u.; per ton of product, the heat
(61) required for this reaction was (see 33):

$$69 \times 5,760 = 397,000 \text{ B.t.u.}$$

The dry top gases carried off the following amount of heat per ton of product (The average top temperature was 325° F.; (see (1) for the gas analysis and (33) for the weight of gas).

	Gas analysis ×	Weight of gas ×	Top temperature ×	Spec. Heat =	B.t.u.
CO_2	0.223	9,359	325	0.2169	147,100
CO	0.224	9,359	325	0.2426	165,300
(62) H_2	0.003	9,359	325	3.4090	31,100
CH_4	0.001	9,359	325	0.5930	1,800
N_2	0.549	9,359	325	0.2438	407,100
Total					752,400

The moisture carried out by the top gases entered the furnace at a temperature of 70° F., *i.e.*, with a heat content (see 33) per ton of product of

(63) $$(70 - 32) \times 603 = 22{,}900 \text{ B.t.u.}$$

Since the moisture was heated in the furnace to 212° F. and evaporated, and the resulting steam superheated to 325° F., the heating of the water required

(64) $$(212 - 32) \times 603 - 22{,}900 = 85{,}600 \text{ B.t.u.}$$

The evaporation required

(65) $$964.8 \times 603 = 581{,}800 \text{ B.t.u.}$$

The superheating required

(66) $$0.48 \times (325 - 212) \times 603 = 32{,}700 \text{ B.t.u.}$$

Therefore the total heat carried off with the moisture of the top gases per ton of product amounts to

	B.t.u.	
	85,600	(see 64)
	581,800	(" 65)
	32,700	(" 66)
(67)	700,100	

Heat Balance Sheet for One Ton of Product

Generated			Consumed				
By	B.t.u	Per Cent.	By			B.t.u.	Per Cent.
Combustion C to CO	3,995,000	31.1	Reduction of Fe_2O_3	6,133,300			
Combustion C to CO_2	6,754,000	52.6	Reduction of Fe_3O_4	490,000	6,623,300		
Hot blast (heat content)	2,057,000	16.0	Reduction of MnO		48,400		
Moisture of the air in hot blast (heat content)	42,000	0.3	Reduction of P_2O_5		17,000		
			Reduction of SiO_2		418,500	7,107,200	55.3
			Calcination of carbonates			702,000	5.5
			Dissociation of moisture in blast			397,000	3.1
			Carried off with the iron			1,142,500	8.9
			Carried off with the slag			909,000	7.1
			Carried off with the dry top gases			752,400	5.9
			Carried off with the moisture in top gas			700,100	5.4
			Radiation, cooling water, and unaccounted for			1,137,800	8.8
Total	12,848,000	100.0	Total			12,848,000	100.0

Discussion

JOSEPH W. RICHARDS, So. Bethlehem, Pa.—This paper answers partly the difficult questions which have come up as to why it is not economical to use higher blast temperatures in the smelting of Mesaba ores, such as the high temperatures used in European practices, and the answer is very plain that the ores will not stand the higher temperature. But the paper does not go into detail as to what special difficulties were found with the higher temperature of blast, and I would ask the author to kindly specify them.

W. A. FORBES, New York, N. Y.—A large part of the troubles that furnace men experienced in using high-blast heats in smelting Mesaba ores in the early days of using Mesaba ores, was due to the attempt to use these soft ores on furnaces having the same lines as the furnaces used in smelting the hard Old Range ores. As the result of observation and study, the dimensions of the lower part of the blast furnaces have been changed and the difficulties of the blast-furnace men in operating with a large percentage of Mesaba ore and high-blast temperatures have been largely overcome. The changes I refer to in particular are: (1) steepening the angle of the bosh; (2), decreasing the height of the bosh; (3) increasing the diameter of the hearth.

JOSEPH W. RICHARDS.—Mr. Forbes's answer is quite satisfactory, and the moral which it points is that when the furnace is run with a higher temperature in the smelting zone, the furnace lines which were adapted for the lower temperatures are not necessarily those best suited for the higher hearth temperatures. If enriched blast were used in a furnace, and higher temperatures run, it would be found necessary to further modify the lines of the furnace to meet the new conditions and to get the maximum output with economy.

J. E. JOHNSON, JR., New York, N. Y.—It seems to me this is one of the few papers which could well have been a little longer. The data which Mr. Mathesius has given us, I think should be considered in connection with the paper which Mr. Brassert read before the American Iron and Steel Institute last spring, in which he described how they had obtained remarkable results with furnaces with steep but very short boshes; the angles have been raised to 80°, and the boshes made so short that they no longer appear as a conspicuous part of the shape of the furnace.

The use of high heats involves an increase in the temperature of the hearth, relative to that of the bosh, and that to a certain extent might be expected, because the ratio of expansion of the gas from its tem-

perature at admission to the temperature at which it leaves the bosh is less, of course, in proportion as the temperature at which it enters is higher; that is to say, the greater its volume the greater its temperature at entering, but it is the same at leaving the bosh, because it leaves it at the same temperature in either event.

There are probably other considerations in regard to the characteristics of the shape, which we do not understand, but which have been worked out at South Chicago, to enable them to get these results.

Another factor which bears upon the case is that they have a remarkable coke—the making of coke has been reduced to a science in recent years; they can make a coke of just the characteristics required, neither too hard nor too soft, too porous nor too dense, to suit their conditions, and by that means they cut down the solution loss, and are enabled to make other improvements which they could not use alone. This is one of the cases where by making three or four improvements you can make another operative, which would not be permissible without them.

One feature in the heat balance which Mr. Mathesius gives is that he uses a very high value for the total heat of the slag and the iron, much higher than is quoted in any of the authorities. I have written to ask Mr. Mathesius if he is positive about the correctness of these figures. I hope he is, because if so it will explain a great many things which have not been explained up to now. For instance, why do we take such a large increase in coke for a comparatively small increase in slag volume? It has not been explicable on the basis of the low value of heat of slag used in the past, but is entirely explicable if his figures are right. Some experiments I made in a crude way some years ago seem to indicate his figures are correct, and I hope he will be able to assure us that they have been checked out and are correct; if so it will be of very great value.

Another point I notice is that the solution loss of the coke, figured from the gas analysis, which is undoubtedly correct, is about 25 per cent. of all the carbon. Twenty-five per cent. of the carbon never gets down to the tuyères, and this results in a very great loss of the heat that we could get in the hearth and also of the heat which we could get in the shaft. The effect of solution loss is complicated. I have been making some investigations along that line recently, and find that some of the old ideas are not in accordance with the facts. For instance, it used to be thought that a low top temperature was the sign of a good working furnace; many people put water in the top of a furnace to hold the dust down and control the top temperature with it. The fact is, on the basis of some figures which I think are absolutely incontestable, it can be shown that the less solution you can have under any conditions within reason, the higher the top temperature will be. That coincides with our experience with Old Range ores, when the solution loss was smaller than it is now,

that the top temperatures ranged higher; in other words, the effect of the solution of carbon by the oxygen from the ore is a very decidedly cooling reaction; it cools off the top gases of the furnace very rapidly.

Such data as Mr. Mathesius has published here are very rare and almost beyond price when one is attempting to find out what really goes on inside the blast furnace. I think the operators of the South Chicago Works are to be congratulated in the economy effected, and the paper is deserving of the greatest of praise because it gives detailed information which is literally almost priceless.

Joseph W. Richards.—Referring to what Mr. Johnson speaks of as the abnormally high amount of heat in the iron and in the slag, taking them as 510 B.t.u. per lb. and 900 B.t.u. per lb., my impression is that those are about average values and are not particularly high. They amount to 285 and 500 calories per kilogram, respectively, which are not abnormal.

I am pleased that Mr. Johnson called attention to the high solution loss; in other words, 25 per cent. of the carbon does not get down to the tuyères under these conditions, which is contrary to what were supposed to be conditions of economical working in the olden days. The real point is, that if the furnace works hot enough without burning all the carbon down there, why do you want to burn it? If you can use some of it above in doing direct reduction, and at the same time get all the heat you need down below, you are better off, and you are running the furnace with so much less carbon per ton of pig iron. There is no use in burning all the carbon down at the tuyères, if you are getting heat enough in the hearth without burning it all. I think the high solution loss is exactly an index of the economy of fuel in this furnace.

W. A. Forbes.—Answering Mr. Johnson, I would point out it is not changes in the diameter of the bosh which have resulted in improved conditions; many of our furnaces have the same diameter of bosh on the Mesaba ores as was used on the Old Range ores; it is the changes in the bosh angle, the height of the bosh, and the diameter of the hearth which have been beneficial.

In regard to coke, it is quite true that, due to the introduction of the by-product coke oven, an improved grade of coke is used at our blast furnaces in the Chicago district. At the same time, the two furnaces treated in this paper operate with the same coke as the other blast furnaces at the same plant. It is at those furnaces which have had their lines modified where the best results are obtained with high blast temperatures with the Mesaba mixtures.

In regard to the top temperature, the reason that water, in our practice, is added to the stock entering the top of the furnace, is primarily to reduce the amount of dust that is blown over with the gases.

H. P. Howland,* South Chicago, Ill. (communication to the Secretary†).—The paper is well worthy of our careful study, not only because of the interesting data in Table I, but more particularly because it brings up the matter of correct stove design.

Table I shows that at South Works they have been able to operate on remarkably low coke and high heats. The question of low coke operation discussed from the cost standpoint is a large one. Where the furnace under discussion is so located that the surplus gas has a large value, the saving in lowering the coke, effected by raising the heat, is one that needs careful consideration. The only way to get the full benefit of this saving is to bring the gas-consuming apparatus to the very highest efficiency.

Mr. Mathesius rightly, therefore, follows his table showing their remarkably low coke consumption by a discussion of stove design. To reap the full benefit of the former, the latter must be much different from that of to-day.

Let us assume two furnaces using washed gas and stoves of 50 per cent. efficiency:

Furnace A: Coke consumption, 1,700 lb.; blast heat, 1,200°; gas, 82 B.t.u.

Furnace B: Coke consumption, 2,100 lb.; blast heat, 800°; gas, 92 B.t.u.

Neglecting the saving made because of the lower amount of blast needed by the low-coke furnace, the heat value of the gas per ton of iron is distributed as follows:

Total B.t.u. in gas	A =	9,922,000
Total B.t.u. in gas	B =	13,800,000
B.t.u. consumed to heat blast:	A =	3,534,000
	B =	2,680,000
Percentage of gas used on stoves:	A =	35.7
	B =	19.4
Surplus B.t.u. for power:	A =	6,388,000
	B =	11,120,000

Power lost by low coke on A: 4,732,000 B.t.u.

Saving by low coke on A is, therefore, 400 lb. of coke at $5 per gross ton = 89c.

Loss by low coke: Assume 1 lb. coal = 11,000 B.t.u. 4,732,000 B.t.u. = 430 lb. coal, at $2 net ton = 43c.

Actual saving of low coke practice: 89 − 43 = 46c.

These figures, of course, only apply to a district where the coke and coal prices are about as above.

There are probably many incidental savings aside from the above that can be attributed to the low coke consumption, such as greater daily tonnage, greater tonnage on lining, better furnace practice in general and a more uniformly working furnace.

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* Superintendent of Blast Furnaces, Wisconsin Steel Co.

† Received Feb. 13, 1915.

However, to gain the full benefit of our low coke consumption, the 43c. loss, due to higher coal consumption, must be eliminated. This can only be done by making the gas from Furnace *A* do the same amount of work as that from Furnace *B*. In other words, we must increase the efficiency of our gas-consuming apparatus.

If we increase our stove efficiency from 50 to 75 per cent. we will use 2,356,000 B.t.u., or 23.7 per cent. of the gas. This leaves for power 7,566,000 B.t.u., which must do the work done on the *B* Furnace by 11,200,000 B.t.u. We, therefore, must increase the efficiency of our power-generating apparatus 50 per cent., as, for example, from 8 per cent. to 12 per cent.

From our experience in developing the efficiency of our old stove equipment at this plant, and from what we are continually learning on the topic, it does not seem to me the day is far off when stoves will be so built as to give us an efficiency of 75 per cent. and give it continuously. There is unquestionably a chance for much increased efficiency along the line of power-generating apparatus. It seems to me, therefore, that we blast-furnace superintendents are in a position to say that if the day approaches of high heats and low coke consumption, we will be able to turn the whole of the saving into dividends. There will then be the added advantage that if our furnaces for any cause should again have higher coke consumption, under the increased stove efficiency, the loss will not be as great.

From my own observation, I would say that there is less reliable knowledge and data about the use of blast-furnace gas than upon any other topic of equal importance in the steel business. Few blast-furnace men know what their stoves are doing, aside from the fact of the heats they are getting. This is due to the fact that when on high coke consumption our gas has been so high in heating value and of such quantity as to go far toward meeting the plant demands. As long as this condition obtained, it was perhaps natural that not much improvement would be shown. Such was the condition at this plant. When, however, we began to operate on our present basis, namely, one furnace instead of three, and the coke consumption on this one in the neighborhood of 1,700 lb., the coal smoke from our boiler house, as well as the monthly steam cost, was a very forcible reminder that not all of the coke saving was net saving.

The natural result was an immediate attempt to improve the efficiency of our stoves. We were fortunate in having a gas-washing plant, the design of which had been changed while relining the furnace, enabling us to wash the gas for the stoves and use unwashed gas on the boilers. As Mr. Mathesius points out, washed gas makes possible the design of a stove from a heat-economy standpoint and not, as was heretofore compulsory because of dirty gas, from the standpoint of easy cleaning.

To me this seems its greatest value, though nearly as great value I would place upon the fact that with the washed gas we are able to measure its volumes accurately, and know much more exactly what we are accomplishing. It is almost impossible to get accurate data on the use of dirty gas.

For the last three months we have been running some tests upon the efficiencies of the stoves. We have made about 50 tests that we consider accurate, these covering several different conditions, as, for instance, five-stove operation, four-stove operation, and attempted three-stove operation. These tests cover operation on atmosphere burners and burners using forced air.

We feel well repaid, both in knowledge and money, for all we have done. When we started the tests there were five stoves in operation: four central-combustion and one side-combustion chamber. The former were small inadequate stoves; the latter a new one with 9-in. checkers. The central-combustion stoves gave an efficiency of 40 per cent. and the other 60 per cent.

By the use of checkers in the combustion chambers the efficiency of two of the small stoves has been increased to 50 and 57 per cent., respectively. We have now eliminated one small stove and are using four stoves, having a combined heating surface of about 130,000 sq. ft. On the two small stoves of 50 and 57 per cent. efficiency, we are using pressure burners. We hope by installing this burner upon the stove of 60 per cent. efficiency to use only three stoves. If this is accomplished, and we are confident it can be, the gas will be burned in a stove plant consisting of three stoves of 50, 57 and 60 per cent., or a combined efficiency of 56 per cent. We were previously using four stoves of 40 per cent. and one of 60 per cent., or a combined efficiency of 44 per cent. We will, therefore, increase our stove efficiency 27 per cent., and consequently use 27 per cent. less gas on our stoves. During the period of these stove tests, covering the last four months, October, November, December, 1914, and January, 1915, the practice on this furnace closely approximates the following figure:

Daily tonnage	564
Pounds of coke per ton	1,700
Cubic feet of air per minute at 62° F. =	36,000
Blast heat	1,100° F.
B.t.u. in washed gas	82

It seems to me that the furnace superintendent with a low-coke furnace must work along such lines as these. It should not be a question so much of tearing down old stoves and building new ones, as an attempt to increase the efficiency of the present equipment, and thereby develop the proper design of the stove of the future. The problem is a very different one from that of simply erecting a new stove plant of larger

capacity of the old design, for we must secure not only the higher blast heat, but higher stove efficiency.

In our use of the pressure burner, we have found that it, in itself, does not necessarily add to the efficiency of the stove, and, consequently, does not necessarily result in any gas saving. The question of gas saving is, and always will be, decided by the efficiency of the stoves, as regards ability to absorb and give up heat. If it is then desired to reduce the number of stoves, the pressure burner will be most essential.

This burner is, however, of great value for two principal reasons:

(1) Accurate mixing of air and gas, with consequent complete combustion.

(2) The large amount of gas that can be forced into a stove. This means that if we have stoves of high enough efficiency we can run on two stoves.

When designing a new stove plant on the basis of washed gas, the following points are most essential: (1) Small checkers of proper design. (2) Large ratio of heating surface to shell radiating surface. (3) A well-insulated shell. (4) A burner designed to force a large amount of air and gas into a stove and mix it correctly.

There is no reason why stoves with 75,000 sq. ft. of heating surface cannot be designed. This stove properly insulated should give an efficiency of 75 per cent.; 11,000 cu. ft. of 82 B.t.u. gas, burned in one such stove per minute for 60 min., will heat 36,000 cu. ft. of air to 1,125° F. for 60 min. The question of high heats will then be simply a question of forcing more gas into the stove. The gas consumption on the above basis is only about 23 per cent. of the gas produced by a 550-ton furnace running on 1,700 lb. of coke.

We have been able to burn 9,000 cu. ft. of gas per minute in one of our small stoves, on a rather temporary pressure burner equipment. This is easily twice what this stove used with the old type of burner.

The question of low coke consumption and high heats is going to bring to our attention very forcibly this matter of efficient use of blast-furnace gas. There certainly is room for great improvement along this line and it would not take much of a prophet to foresee the entire remodelling of our stove equipment in the next ten years.

Walther Mathesius, South Chicago, Ill. (communication to the Secretary*).—Referring to Mr. Johnson's question as to the amount of heat carried off with the molten metal and the slag, I beg to say that the figures used in my heat balance represent average values as generally quoted in German literature (cf. Ledebur: *Handbuch der Eisenhuettenkunde*, vol. ii, p. 258, 1906).

In order to determine experimentally these figures I made a number

* Received Mar. 25, 1915.

of calorimetric tests some years ago while working along similar lines of research at German blast-furnace plants. In nearly all cases I found that on regular foundry and Bessemer grades the results of my tests so closely approached the average values given in literature, that for the purpose of a general heat balance I felt justified in uniformly using the latter figures. Since neither chemical analysis nor the apparent physical condition of the iron and slag at Bessemer furnaces using Mesaba ores is decidedly different from the German practice on similar grades, I have no doubt that for the purpose of a general heat balance the German figures for the amount of heat in the iron and slag can be taken as representative also of American conditions.

Modern Gas-Power Blower Stations

BY ARTHUR WEST, SOUTH BETHLEHEM, PA.

(New York Meeting, February, 1915)

It is the purpose of this paper to describe briefly some recent large power stations for blast furnaces, where the blast is exclusively supplied by gas engines using furnace gas. The stations are given in the chronological order in which they were designed, in order to indicate the various improvements suggested by experience.

Fig. 1 shows the blowing-engine station of the Bethlehem Steel Co. at Bethlehem. Although the picture shows only five blowing engines, the house now contains 11 engines. In the diagrammatic plan the arrow shows the position and direction of camera. This station handles five modern blast furnaces of 450 to 500 tons capacity each. In this, as in all the other stations described in this paper, each furnace is blown by two single tandem gas engines. Ten engines are thus required for five furnaces, leaving one single tandem spare engine. This spare capacity has been shown by several years' experience at Bethlehem to be adequate to insure continuous operation. The engines in the Bethlehem station are right hand and left hand, arranged in pairs. The outboard bearings of each pair are close together so that there is not room for passageway between them. On the right of the picture are shown the inclined drums communicating with the cold-blast lines outside of building, so that any engine may be operated on any furnace. The illustration shows four cold-blast gate valves with pipes passing through wall, but since the photograph was taken the drums have been extended to take care of a fifth valve for a fifth furnace.

At the extreme back of the picture can be seen the tubs of three twin vertical-horizontal Southwark steam blowing engines, which, although of modern design and as good as new, are not operated because of the greatly reduced blowing cost shown by the gas blowing engines.

The next station built is shown in Fig. 2. This is the plant of the Minnesota Steel Co. at Duluth, Minn., and is at present constructed for five engines, to handle two furnaces, with one spare engine. The engines are of practically the same size as those at Bethlehem. Instead of being arranged in right-hand and left-hand pairs, as at Bethlehem, all the engines are of the same hand; this arrangement has some ad-

vantage over the Bethlehem plan, in that it allows passageway between adjacent engines instead of only between alternate engines. The Minnesota plan, however, requires a longer house for the same number of engines than does the Bethlehem plan. There is no special operating advantage in having the engines all of one hand, because these engines are designed so that all parts subject to renewal are independent of the hand of the engine.

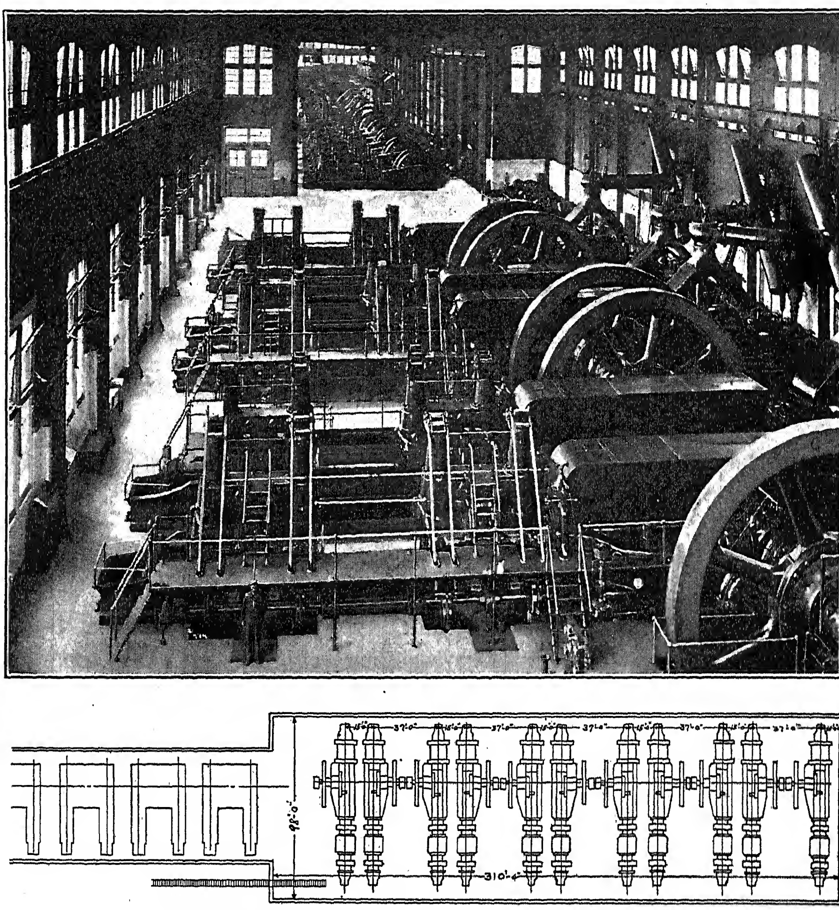


FIG. 1.—BLOWING-ENGINE STATION OF BETHLEHEM STEEL CO., BETHLEHEM, PA.

This station has been completed some months, but has never been run, as the new steel plant is not yet in operation. No steam blowers have been provided to blow in the new furnaces; the blowing engines will be operated on gas from the open-hearth producers until blast-furnace gas becomes available.

Figs. 3 and 4 illustrate the blower station of the Maryland Steel Co. at Sparrows Point, Md. These engines are the same size as those of

the Minnesota Steel Co. As the photographs and plan view show, the engines are arranged in right-hand and left-hand pairs, as at Bethlehem. In this case, however, the engines are set diagonally in the house. It will be seen that with this construction the outboard crank-shaft bear-

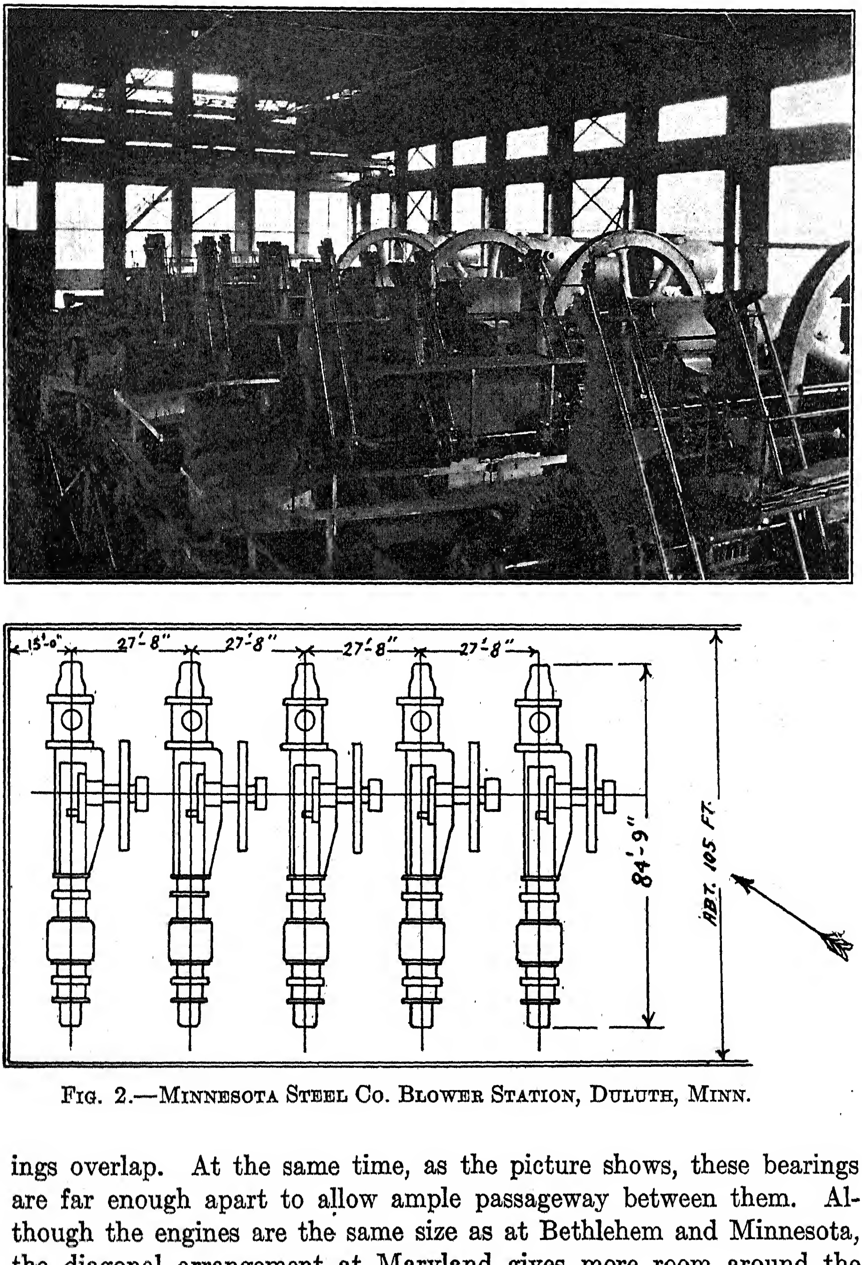
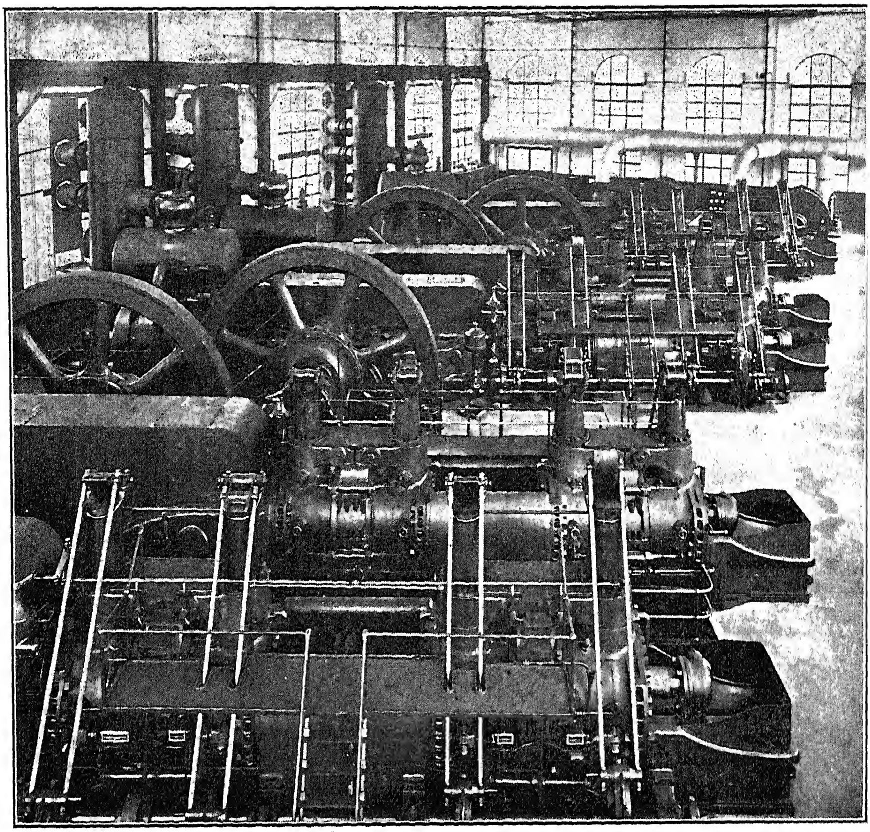


FIG. 2.—MINNESOTA STEEL CO. BLOWER STATION, DULUTH, MINN.

ings overlap. At the same time, as the picture shows, these bearings are far enough apart to allow ample passageway between them. Although the engines are the same size as at Bethlehem and Minnesota, the diagonal arrangement at Maryland gives more room around the



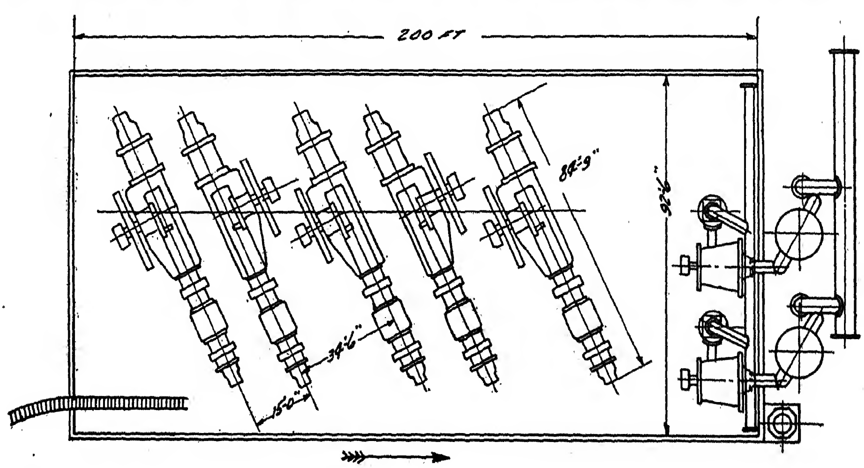


FIG. 3.—MARYLAND STEEL CO. BLOWER STATION, SPARROWS POINT, MD. LOOKING TOWARD THEISENS.

engines, with a power house 20 ft. narrower. The length of the house is not increased, both because the outward bearings lap by each other, and because room remains in the triangular space at the end of the station for a railroad car under the traveling crane. With the engines set squarely across the house, at least one extra panel must be provided to allow the entrance of such a car. The power house at Maryland being 20 ft. narrower than at Bethlehem and Minnesota is, of course, less expensive, both because the area of the power station is

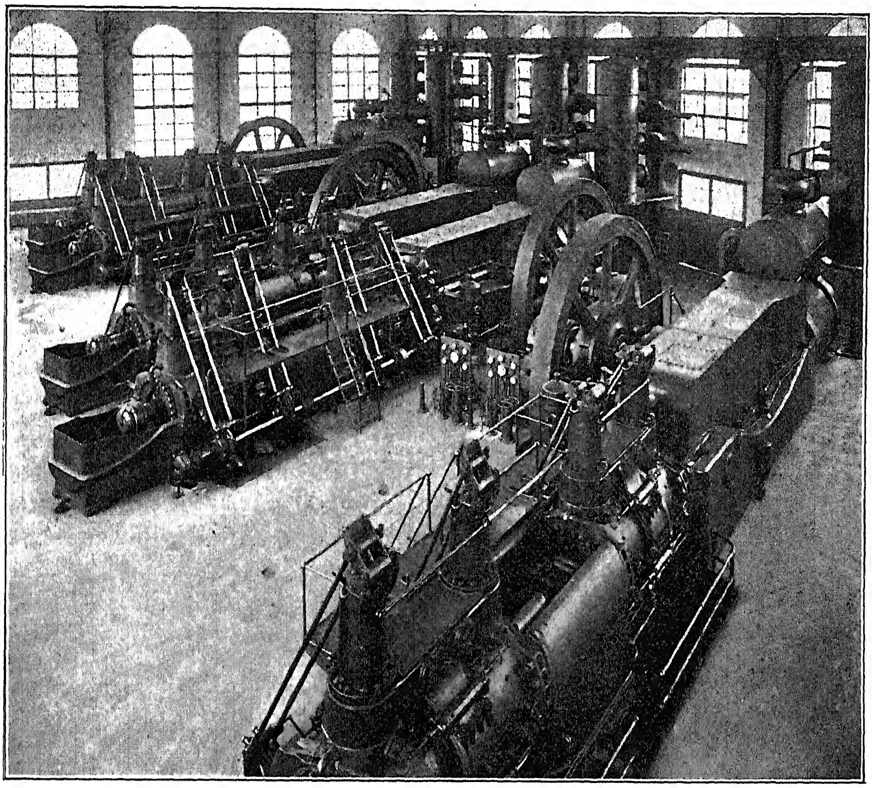


FIG. 4.—MARYLAND STEEL CO. BLOWER STATION. LOOKING AWAY FROM THEISENS.

less and because the cost of the building per square foot is less, the cost of each roof truss and of the traveling crane being reduced because of the lesser span. As shown in the illustration, the gauge boards and operating apparatus are brought together for a pair of blowing engines. The cold-blast valves and snorting valve on each blower are operated by compressed air, controlled from in front of the gauge board. Without moving from the front of the gauge board, one man single handed can put two engines on a furnace very quickly. The operations are:

1. By means of compressed air, open cold-blast valve to proper furnace, thus letting the blast pressure directly on to the Dyblie automatic check valve shown on top of tub.
2. By same means, open air snorting valve shown at side of tub.
3. Open jacket-water valve. Water will be seen discharging into funnel shown at rear of tail slide of engine.
4. Throw in igniter switch.
5. Open gas throttle.
6. Open valve supplying compressed air for starting gas engine. The engine will immediately pick up speed and run under the control of the governor, the tub discharging its air through the snorter out of doors, the Dyblie valve remaining seated under the blast pressure.
7. By means of compressed air, close snorter valve. The Dyblie valve automatically opens wide and the engine is blowing the furnace. The engine being all the time under the control of the governor, its speed does not vary when it is put onto the furnace. It being quite unnecessary to warm up as in the case of a steam engine, such a gas engine can be put on a furnace very rapidly, always inside of 1 min. I have seen it done in 35 sec., the engine being absolutely cold. It is unnecessary to adjust any oil feeds on the engine as all oil supply automatically stops and starts with the engine.

This station contains five engines for two furnaces, but is designed to be lengthened to allow the installation of four more engines, thus providing nine engines for four furnaces. As shown, the blast drums are provided with four openings for four furnaces.

Fig. 5 shows a nearer view of the speed-governing apparatus. H. A. Brassert, in his paper before the American Iron and Steel Institute on Modern American Blast Furnace Practice, stated that "*uniformity* of conditions is the keynote of successful furnace operation." He also calls attention to the fact that of all the elements introduced into a furnace in a given time, air is the greatest, both by volume and by weight. It is evident, therefore, that the air should be measured into the furnace as carefully and as uniformly as are the ore, coke, and limestone. Now, a good modern blowing tub with automatic valves is, outside a holder, the best known means for measuring air. With properly designed air valves, leakage is infrequent. If such leakage exists, it can be detected by ear immediately and can be easily remedied. Since the production capacity of the blast furnace, per unit of time, must be held constant, even though the blast pressure varies over wide limits, it follows that the weight of air delivered to the furnace per minute must also remain constant no matter what the variation in blast pressure. For this reason all the blowing engines described in this paper have air

tubs designed with the utmost care. In addition, the engines are equipped with speed governors as sensitive as those employed on the same gas engines operating on 60-cycle current in parallel. In Fig. 5 is shown such a governor equipped with means for holding the engine at any desired speed, irrespective both of fluctuations of the blast pressure and of heat value of the furnace gas. Without such care in the governing apparatus, a change in either or both of these conditions will cause a change of speed and a consequent variation in the weight of air as compared to the weight of coke charged to the furnace. This is one of the foes to that uniformity desired by all successful furnace managers.

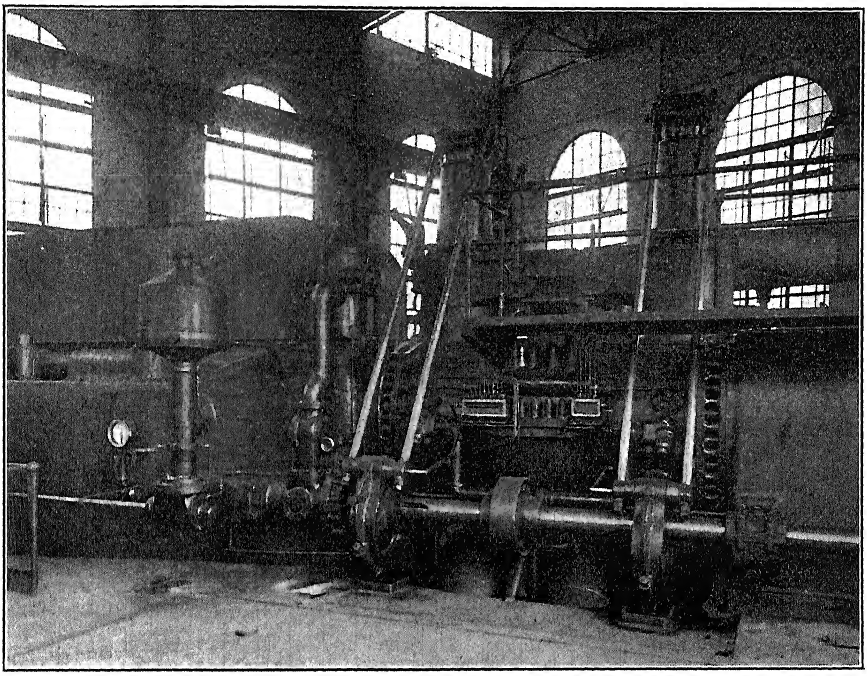


FIG. 5.—MARYLAND STEEL CO. BLOWER STATION. NEAR VIEW OF GOVERNOR.

To the left of the governor is shown a sensitive recording speed chart. This shows just how much air per minute was being charged to the furnace at any given time. Incidentally, it also gives the exact time and duration of furnace checks.

In all the furnace-gas power plants of which the writer has knowledge, either in this country or in Europe, it has been the practice to place the apparatus for cleaning the engine gas far from the engine house. When, however, the Maryland Steel station was being laid out, the writer recommended the installation of the Theisen washers (Fig. 6)

directly in the engine room. Since the economical operation of the gas engines is dependent upon the cleanliness of the gas, it seems best to charge the chief engineer of the power station directly with the control of and responsibility for the engine gas-cleaning apparatus. This arrangement has proved very successful at the Maryland plant. It only required the addition of one bay to the length of the station and saved nearly the whole first cost of the Theisen house, with its traveling crane, etc. It also concentrates responsibility and saves operating labor, besides being agreeable to the safety department.

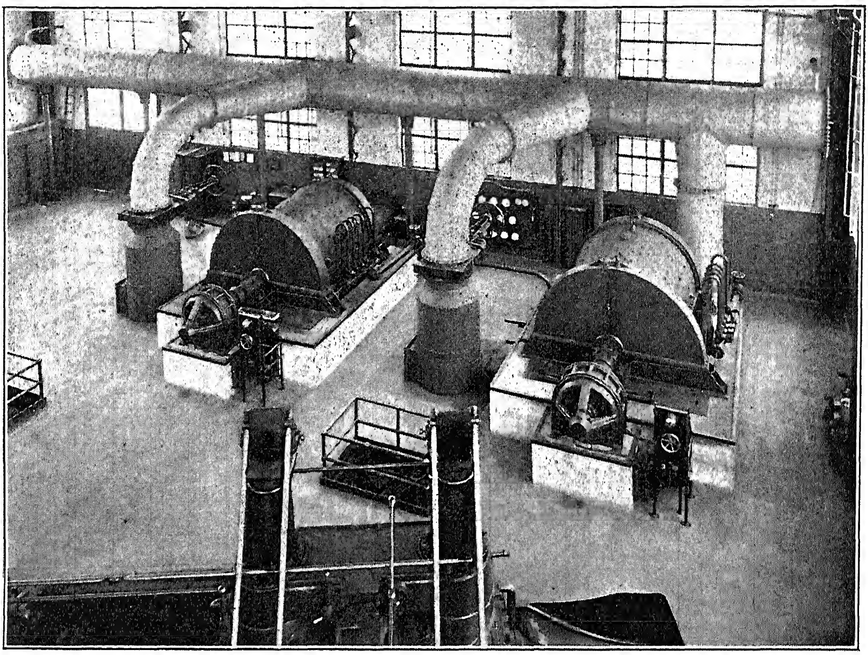


Fig. 6.—Theisen Washers at the Maryland Steel Co. Blower Station.

Each of the Theisens shown cleans gas enough for five gas engines, or more than enough to blow two furnaces. With the Maryland station of its present size, one Theisen is operated, leaving the other as a spare. When the station is enlarged to care for four furnaces, a third Theisen will be installed in the vacant space to the left. Two Theisens will then be regularly operated, leaving the third as a spare.

Fig. 7 shows the outside appearance of the Maryland blower station. The pipe on the right brings raw gas to the station; the gas having been cleaned only so much as is required for the stoves and boilers. Two towers can be seen just outside the station at the right, each tower being in series with each Theisen. The tower is empty and is provided with water spray, up through which the gas passes on its way to the Theisen.

The tower does some cleaning, but should be regarded principally as a cooler.

The Theisens clean the gas very effectively and are not at all extravagant in their consumption of power, which is about 2 per cent. of the power of the engines they supply. A new type of disintegrator, said to consume very much less power and water than the present Theisen apparatus, is in use in Europe.

At the right of the picture is seen the exhaust stack, which connects to an exhaust tunnel, incorporated in the concrete foundation of the building. A certain amount of salt water is discharged into this exhaust tunnel to keep the temperature below 212°. Such an exhaust tunnel is a perfect silencer and is in successful use at Gary, Bethlehem, and Sparrows Point.

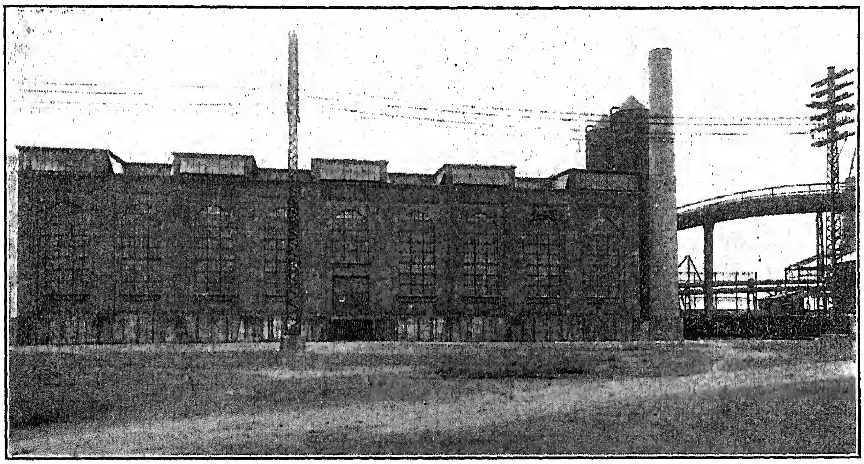


Fig. 7.—Outside View of Maryland Steel Co. Blower Station.

The station of the Maryland Steel Co. is also provided with a cooling pond, located just to the right of the exhaust stack and in line with the power station. This was provided because only brackish water was available, which it was not considered desirable to use in the gas engines. The cooling pond has no towers in connection with it and works very satisfactorily, requiring the operation of only one centrifugal pump in addition to the apparatus that would otherwise be required. In cases where water is very bad or very scarce, such a gas station with a cooling pond will furnish power much more cheaply and simply than can any steam plant. In fact, a careful examination of the Maryland plant will convince an experienced blast-furnace manager that such a gas-power station is not only much more economical than a steam station, but is much simpler; the boiler plant, condensing apparatus, etc., being entirely eliminated.

Effect of Finishing Temperatures of Rails on Their Physical Properties and Microstructure

BY W. R. SHIMER, SO. BETHLEHEM, PA.

(New York Meeting, February, 1915)

IN his valuable report on Finishing Temperatures and Properties of Rails,[1] Dr. G. K. Burgess, Chief of the Division of Metallurgy, U. S. Bureau of Standards, has begun a line of investigation which should be continued by those interested in the subject, and who have proper facilities for carrying out the work. For the past year or more the Bethlehem Steel Co. has been conducting experiments to learn the effect of rail finishing temperatures on their physical properties and microstructure and the results are here given for the consideration of those interested in the rail situation.

Considerable differences of opinion have been expressed by various authorities, in the past, as to the effect of large or small grain size and high or low finishing temperatures of steel rails on their physical properties and wearing qualities. Low finishing temperatures and, therefore, low shrinkage, have been advocated on the theory that the wearing qualities would be improved. Others have recommended high finishing temperatures.

The generally accepted cause for rail failures, due to the development of transverse fissures, is excessive wheel pressure on hard rails. One of the causes advanced for hard rails has been that they contain too high percentages of carbon and manganese; another that the rails were finished at too low a temperature, *i.e.*, at or near the critical point.

In the experiments herein described the rails were rolled from reheated blooms. All the rail blooms were charged hot in a reheating furnace and brought up to about the original ingot-rolling temperature before rolling into rails. These rails gave better results in deflection, withstood a greater number of drops before breaking, and showed greater ductility than rails of identically the same composition and section which were rolled direct from the ingot. Rails rolled from reheated blooms, finished at somewhat higher temperatures than when rolled direct from the ingot, have a different microstructure, which the writer does not believe is due so much to the difference in finishing temperatures, as to the difference in

[1] *Technologic Paper No.* 38, *N. S. Bureau of Standards* (1914); *Trans.*, l, p. 302, (1914).

reduction in one heat. For example, a direct-rolled rail is reduced from the ingot to the rail in one heat, whereas when the blooms are reheated, all the strains set up in the reduction of the ingot to the bloom are relieved during the reheating process.

A few results of drop tests on rails of identical section and of approximately the same composition are given, comparing rails rolled direct with those rolled from reheated blooms:

Heat A.—100-lb. Rails Rolled Direct

C, 0.600; Mn, 0.64; Si, 0.068; P, 0.018; S, 0.032

Ingot No.	No. Drops	Permanent Set, Inches	Elongation						
			1 in.	2 in.	3 in.	4 in.	5 in.	6 in.	Total
2	1	1.08	0.05	0.07	0.06	0.05	0.05	0.04	0.32
11	1	1.07	0.05	0.06	0.07	0.05	0.05	0.04	0.32
	2	2.02	0.10	0.11	0.11	0.10	0.09	0.06	0.57
	3	4.06	0.13	0.14	0.15	0.13	0.12	0.12	0.96
	4	Broke	No	t meas	ured				
19	1	1.07	0.06	0.06	0.06	0.05	0.05	0.03	0.31

Average deflection after the first drop, 1.07 in.

Heat B.—100-lb. Rails Rolled from Reheated Blooms (Compare with Heat A)

C, 0.600; Mn, 0.66; Si, 0.080; P, 0.018; S, 0.034

Ingot No.	No. Drops	Permanent Set, Inches	Elongation						
			1 in.	2 in.	3 in.	4 in.	5 in.	6 in.	Total
3	1	1.60	0.04	0.05	0.05	0.05	0.04	0.03	0.26
	2	2.90	0.08	0.10	0.11	0.11	0.09	0.07	0.56
	3	4.15	0.10	0.13	0.16	0.17	0.15	0.11	0.82
	4	5.40	0.11	0.14	0.17	0.19	0.16	0.12	0.89
	5	Broke	0.11	0.14	0.18	0.19	0.17	0.14	0.93
2	1	1.60	0.04	0.05	0.05	0.05	0.04	0.03	0.26
1	1	1.40	0.05	0.06	0.07	0.07	0.06	0.03	0.34

Average deflection after the first drop, 1.53 in.

Heat C.—100-lb. Rails Rolled Direct (Compare with Heat D)

C, 0.646; Mn, 0.81; Si, 0.105; P, 0.028; S, 0.029

Ingot No.	No. Drops	Permanent Set, Inches	Elongation						
			1 in.	2 in.	3 in.	4 in.	5 in.	6 in.	Total
2	1	1.30	0.03	0.03	0.05	0.05	0.05	0.03	0.24
	2	2.30	0.04	0.06	0.08	0.09	0.09	0.08	0.44
	3	3.30	0.07	0.09	0.13	0.15	0.14	0.11	0.69
Middle	1	1.20	0.02	0.03	0.04	0.05	0.04	0.04	0.22
	2	2.30	0.06	0.07	0.08	0.09	0.08	0.07	0.45
Last	1	1.20	0.03	0.04	0.05	0.05	0.04	0.03	0.24

Average deflection after the first blow, 1.23 in.

Heat D.—100-lb. Rails Rolled from Reheated Blooms

C, 0.648; Mn, 0.83; Si, 0.081; P, 0.028; S, 0.031

Ingot No.	No. Drops	Permanent Set, Inches	Elongation						
			1 in.	2 in.	3 in.	4 in.	5 in.	6 in.	Total
1	1	1.45	0.04	0.04	0.06	0.05	0.04	0.03	0.26
	2	2.60	0.06	0.08	0.12	0.09	0.09	0.05	0.49
	3	3.70	0.10	0.13	0.17	0.15	0.12	0.09	0.76
	4	4.70	0.10	0.13	0.17	0.17	0.15	0.11	0.83
	5	Broke	0.12	0.14	0.18	0.18	0.15	0.11	0.88
2	1	1.45	0.06	0.07	0.07	0.06	0.04	0.03	0.33
3	1	1.50	0.04	0.05	0.06	0.05	0.04	0.03	0.27

Average deflection after the first drop, 1.47 in.

Heat E.—100-lb. Rails Rolled Direct (Compare with Heat F)

C, 0.740; Mn, 0.78; Si, 0.104; P, 0.020; S, 0.033

Ingot No.	No. Drops	Permanent Set, Inches	Elongation						
			1 in.	2 in.	3 in.	4 in.	5 in.	6 in.	Total
2	1	1.01	0.04	0.05	0.03	0.03	0.02	0.02	0.19
	2	1.90	0.05	0.06	0.07	0.08	0.05	0.05	0.36
10	1	1.01	0.04	0.04	0.04	0.04	0.02	0.02	0.20
19	1	1.01	0.03	0.04	0.05	0.05	0.04	0.03	0.24

Average deflection after the first drop, 1.01 in.

Heat F.—100-*lb. Rails Rolled from Reheated Blooms*

C, 0.750; Mn, 0.81; Si, 0.071; P, 0.018; S, 0.029

Ingot No.	No. Drops	Permanent Set, Inches	Elongation						
			1 in.	2 in.	3 in.	4 in.	5 in.	6 in.	Total
1	1	1.50	0.04	0.05	0.05	0.05	0.03	0.03	0.25
	2	2.60	0.06	0.09	0.11	0.10	0.09	0.07	0.52
	3	3.70	0.09	0.12	0.15	0.15	0.14	0.09	0.74
	4	4.80	0.10	0.15	0.18	0.18	0.15	0.10	0.86
	5	6.00	0.12	0.16	0.19	0.19	0.16	0.12	0.94
			Not	tested	to des	tructi	on		
2	1	1.40	0.06	0.06	0.07	0.07	0.05	0.04	0.35
3	1	1.30	0.02	0.04	0.05	0.05	0.04	0.03	0.23

Average deflection after the first drop, 1.40 in.

The drop-test results of the rails rolled from reheated blooms were superior to those of rails rolled direct from the ingot.

Numerous records of direct-rolled rails show the results of drop tests from the same heat to be not uniform with test pieces of uniform chemical composition, taken from the same position in the ingot and free from segregation.

Drop tests of rails rolled from reheated blooms show consistently uniform results under the same conditions.

The reason rails rolled direct do not show uniform drop-test results seems to be that, when two or three blooms are rolled from an ingot, rails rolled from the first bloom are finished at the highest temperature, rails rolled from the second bloom are finished at the next highest temperature, while the rails rolled from the last bloom of the ingot are finished at a considerably lower temperature than the first two. If there are any delays at the mill and any blooms are held up they sometimes become too cold to roll, and if they are just hot enough to roll, the rails will be finished below the average temperature.

In the case of rails rolled from reheated blooms a uniform temperature can be obtained on all rails rolled. One bloom is drawn from the reheater at a time and sent through the mill. If there is any delay at the mill the blooms remain in the reheater until the mill is ready to roll them. Any desired temperature can be maintained in the reheating furnace so as to obtain the desired finishing temperature on the rails. The temperature of the reheater is regulated according to the composition and section of the rails being rolled.

Later in this report, results of an investigation on finishing rails at different temperatures are recorded. Nine 19 by 23 in. ingots from the same heat were each rolled into two "three-rail" blooms, each ingot mak-

ing two blooms, or six rails 33 ft. long of 100-lb. section. The composition of this heat was: C, 0.720; Mn, 0.73; P, 0.019; S, 0.037. These 18 blooms were charged direct from the bloomer into the reheating furnace and heated up to the original ingot-rolling temperature and, after drawing from the furnace, some were held for different lengths of time, before rolling, so as to obtain a range of high and low finishing temperatures on the rails rolled from them.

Finishing temperatures were measured on all the rails directly after leaving the finishing rolls. Shrinkages were measured on all the rails. A drop-test piece, 4 ft. long, was cut from each *A*, *B*, *C*, *D*, *E* and *F* rail from each of the nine ingots, making a total of 54 drop-test pieces.

A short piece, 6 in. long, was cut from each rail directly back from where the drop-test pieces were cut. A standard tensile bar (0.505 in. diameter by 2 in. long) was turned up from both sides of the head of these short rail sections—one from the bottom side of the head of each section as it cooled on the bed, and one from the top side—making 108 tensile bars.

Next to the piece for tensile test a thin section was sawed from the head of each of the 54 rails, and used for making Brinell hardness tests.

Finishing temperatures ranging from 2,147° to 1,652° F. were obtained in this experiment.

For convenient study, the results of drop tests, physical tests, Brinell tests and check carbons on all the rails are given in the accompanying table. In the table are given the minimum deflection, or the deflection after the first blow of the drop; the maximum deflection, or the last deflection measured before the drop-test piece broke; the number of drops of the tup required to break the rail; the total elongation measured in 6 in., after the rail broke; the tensile results of the bar cut from the top side of the head of the rail as it cooled on the bed; the tensile results of the bar from the bottom side of the head as it cooled on the bed.

Ingots were numbered from one to nine, inclusive. The positions of the rails in each ingot were as follows: *A* the top rail of the ingot, *B* the second from the top, and so on down, *F* being the bottom rail of the ingot. The *A*, *B*, and *C* rails of all ingots were rolled from the upper bloom; the *D*, *E*, and *F* rails were rolled from the bottom bloom of the ingots.

The shrinkages recorded are averaged from measurements of the three rails rolled from each of the 18 blooms.

The finishing temperatures are fairly consistent with the shrinkages, but there are a few cases where they are not quite so, due to the difficulty experienced by the operator of the optical pyrometer in matching the color correctly.

A microscopic examination was made of the threaded end of each of the 108 tensile bars. The micrographs shown at the end of this report reproduce the average structure of the top and bottom tensile bars from

the 6-in. rail pieces from the nine ingots. All micrographs are magnified 100 diameters.

Drop Tests.—All rails were tested head up. Six 1-in. spaces were marked with a center punch on the base of the test piece, and elongations were measured from these punch marks after the rail broke. In some cases the rails bent almost at right angles without breaking in the base; in these cases the elongations are not as great as they would have been had the rail broken in the base. Figs. 10 and 11 show some of the test pieces which bent to almost 90° without breaking in the base, proving them to be exceedingly ductile, although the ductility measurement does not indicate this. The height of drop was 18 ft. and the weight of the tup 2,000 lb.

The following summary is drawn from a practical standpoint based on the results of the various tests and microscopic examinations and a study of numerous records.

Summary

No appreciable difference in grain was found to indicate that the size or structure was governed by a difference in finishing temperature.

There was practically no free ferrite in any of the *A* rails, slightly more in the *B* rails, more in the *C* rails, and so on; the *E* and *F* rails from all the nine ingots contained an almost complete network of free ferrite.

The *A*, *B*, *C*, and *D* rails all had a more or less sorbitic structure, and the *E* and *F* rails were not sorbitic. The only explanation the writer can offer for this consistent difference between the microstructures of the various rails according to their position in the ingot is as follows:

(1) The absence of free ferrite in the *A*, *B*, and *C* rails seems to be due partly to the presence of a consistently higher percentage of carbon than in the *D*, *E*, and *F* rails.

(2) When these experimental rails were rolled, the rails from each ingot were kept apart on the bed to avoid their being mixed. For example: When Bloom 1*ABC* was rolled, the *A* rail was sawed to length first, and the *C* rail last. When they were put on the bed, the *C* rail was first, the *B* rail next with its base resting against the head of the *C* rail, and the *A* rail last with its base against the head of the *B* rail. These rails were cooling at least 3 to 5 min. before the *D*, *E*, and *F* rails from the bottom bloom of No. 1 ingot came along. When the *D*, *E*, and *F* rails went over to the bed, the *F* rail went first, a small space being left between it and the *A* rail; the base of the *E* rail rested against the head of the *F* rail, and the base of the *D* rail rested against the head of the *E* rail. The head of the *D* rail was left exposed. Therefore, the rails from the various ingots cooled at different rates of speed. The heads of all the *A* rails were air cooled and the structure was sorbitic. The heads of the *B* and *C* rails cooled more slowly on account of having hot rails resting against them, consequently these rails contained more free ferrite than the *A* rails. The

C rails contained more ferrite than the *B*, and the *B* more than the *A* rails. The head of the *C* rails had an annealing effect longest, the *B* next, and the *A* rail none.

Now, as to the *D*, *E*, and *F* rails. The *F* rail was the first to have a hot rail resting against its head, the *E* rail next, while the head of the *D* rail was air cooled. This, it seems to me, shows that this consistent difference in the separation of free ferrite, according to the position of these rails in the ingot, is due simply to the difference in the rate of cooling of the rails.

In order to learn what would happen if samples of *A*, *B*, *C*, *D*, *E*, and *F* rails from the same ingot were all annealed at the same temperature and cooled at the same rate of speed, samples from rails 1*A*, 1*B*, 1*C*, 1*D*, 1*E*, 1*F*, 9*A*, 9*B*, 9*C*, 9*D*, and 9*E* were charged in the same muffle furnace, heated to slightly above the critical point (1,450° F.), and left to cool slowly in the furnace over night. Micrographs were taken of the structures of these pieces after annealing and are shown in Figs. 12 and 13.

It will be seen that after this annealing, the structures of all the rails, except 1*F*, were about the same, and contained no free ferrite. Rail 1*F* contained a network of free ferrite, but as this sample contained only 0.622 per cent. carbon and the other 10 samples contained no less than 0.680 per cent., the difference in structure of this sample could be due to its lower carbon content. In the other 10 samples the aggregate of ferrite and pearlite has changed to a homogeneous solid solution.

The difference in carbon, together with the difference in the rate of cooling, accounts for the separation of free ferrite in some instances and the non-separation in others.

The above samples were afterward heated to 1,800° F. and allowed to cool slowly in the furnace over night. These samples were polished and etched and micrographs taken of their respective structures. In this case the grain size increased and free ferrite separated out in the boundaries of the pearlite crystals. Approximately the same amount of free ferrite was observed in all the pieces, showing that when heated to a rail-finishing temperature and allowed to cool at the same rate of speed the microstructure is the same throughout, regardless of the position of the rail in the ingot. These micrographs are shown in Figs. 14 and 15.

The average physical results of the 54 tensile bars tested from all the *A*, *B*, and *C* rails are as follows:

Tensile Strength	Elastic Limit	Elongation, Per Cent.	Contraction, Per Cent.
125,130	64,150	14.0	19.12

The average physical results of the 54 tensile bars tested from all the *D*, *E*, and *F* rails are as follows:

Tensile Strength	Elastic Limit	Elongation, Per Cent.	Contraction, Per Cent.
116,300	59,720	16.2	24.54

The average carbon content of the *A*, *B*, and *C* rails is 0.711 per cent.; of the *D*, *E*, and *F* rails, 0.678 per cent.

The *A*, *B*, and *C* rails are three points higher in carbon; higher in tensile strength; higher in elastic limit; and lower in elongation and contraction of area than the *D*, *E*, and *F* rails. Apparently the slight difference in carbon content has caused the difference in physical properties. Check analyses were made for manganese, silicon, phosphorus, and sulphur, and all the results were so close to the original heat analysis that the difference in physical properties cannot be due to these elements.

The following are the average carbon contents and average tensile results (each an average of six tests), finishing temperatures, and shrinkage of the rails rolled from each bloom:

ABC Rails, Ingot	Tensile Strength	Elastic Limit	Elongation, Per Cent.	Contraction, Per Cent.	Carbon, Per Cent.	Finishing Temperature, °F.	Shrinkage, Inches
1	128,000	66,600	14.6	17.91	0.712	2,146	6.60
2	124,330	64,660	12.8	16.90	0.713	2,138	6.69
3	125,910	64,330	13.9	20.20	0.702	2,049	6.64
4	128,000	64,660	14.2	19.04	0.711	2,120	6.77
5	125,660	65,180	11.9	17.39	0.700	2,147	6.64
6	123,660	63,110	14.4	20.56	0.700	2,012	6.37
7	124,500	63,350	14.1	19.84	0.706	1,868	6.06
8	124,500	63,330	16.0	20.30	0.733	1,850	5.85
9	121,600	61,800	14.1	19.98	0.714	1,832	5.66
Average....	125,130	64,150	14.0	19.12	0.711		
DEF Rails, Ingot							
1	118,750	60,500	15.6	24.71	0.669	2,020	6.44
2	116,580	59,660	16.5	23.90	0.664	2,084	6.63
3	116,830	60,670	15.9	23.07	0.667	2,120	6.71
4	116,100	59,500	15.7	22.41	0.650	2,147	6.65
5	116,000	59,500	16.1	24.09	0.676	2,048	6.56
6	116,100	60,400	15.7	22.07	0.665	1,904	6.37
7	115,660	59,500	16.6	25.81	0.674	1,850	5.81
8	115,400	59,250	16.9	24.83	0.681	1,841	5.69
9	115,330	58,910	17.0	30.00	0.680	1,652	5.37
Average....	116,300	59,720	16.2	24.54	0.678		

The above shows that the rails finished at high temperatures have the same strength and ductility as those finished at low temperatures, this conclusion being confirmed by a study of the drop-test results and the micrographs. The latter show that the structure of all the *A* rails is identical; all the *B* rails have the same structure; in short, all rails from the same relative position in the various ingots show the same micro-

structure. The only difference found, regardless of finishing temperatures, was between the physical properties of the *A*, *B*, and *C* rails and the *D*, *E*, and *F* rails, which was dependent upon the position of the rails in the ingot. The difference in microstructure was due to the rate of cooling when the rails were on the bed.

The above results refer to rails rolled from reheated blooms. From past experience it has been noticed that the rails rolled direct from the ingot do show differences in physical properties, varying in accordance with their finishing temperatures. The difference, it seems, is due to the fact that when rolling direct from a 19 by 23 in. ingot, or an 18 by 19 in. ingot, considerable strains are set up during this continued reduction, which are increased or decreased according to the finishing temperature.

When rails are rolled from reheated blooms, all the strains set up during the reduction of the ingot to the bloom are removed during the reheating process, since the blooms are reheated to or near the original ingot-rolling temperature. The percentage of reduction from the bloom to the finished rail is not so great as from the ingot to the finished rail, consequently no such strains are set up and therefore a difference in finishing temperature does not seem to affect appreciably the physical properties and microstructure of rails rolled from reheated blooms.

A careful study of the results of the drop tests recorded in this report shows that the rails finished at high temperatures are equally as good as those finished at the lower temperatures. Figs. 10 and 11 show drop-test pieces representing the following rails:

1-E—Finished at 2,020° F.; shrinkage 6.44 in.; stood six drops and did not break in the base.

1-F—Finished at 2,020° F.; shrinkage 6.44 in.; stood six drops and did not break in the base.

3-B—Finished at 2,048° F.; shrinkage 6.64 in.; stood five drops and did not break in the base.

4-C—Finished at 2,120° F.; shrinkage 6.77 in.; stood five drops and did not break in the base.

5-D—Finished at 2,048° F.; shrinkage 6.55 in.; stood five drops and did not break in the base.

9-B—Finished at 1,832° F.; shrinkage 5.66 in.; stood six drops and did not break in the base.

In addition to those shown in the photographs, 9-D and 9-E finished at 1,652° F.; shrinkage 5.37 in.; stood six drops and five drops respectively and broke in the same manner as those illustrated.

The following drop-test pieces also broke in the manner above described: 1-A, 1-D, 1-E, 1-F, 2-A, 3-A, 3-B, 3-C, 3-D, 4-C, 4-D, 4-E, 5-A, 6-A, 6-B, 6-C, 7-D, 9-B, 9-D and 9-E. These drop-test pieces represent a wide range of finishing temperatures, shrinkage, and microstructure,

but have almost identical physical characteristics as shown by the results of this test, which is the important test of the quality of a rail.

The conclusions above are based on the results of a carefully carried out investigation along practical lines. The primary object of these experiments was to determine the best finishing temperature for steel rails. There was a variation of 500° F. in the finishing temperatures and a variation of 1.34 in. in shrinkage. Even with this wide range in temperature and shrinkage, it was impossible to determine, from the results of the respective tests, at what temperature a rail should be finished in order to obtain the best results, since all the tensile and drop-test results recorded are good, and practically identical. The results show it is equally impossible to decide what microstructure and shrinkage are desirable.

Further, from the results of this investigation we have learned that rails rolled from reheated blooms are more ductile than those rolled direct from the ingot, regardless of their finishing temperature. Also, we have learned to be cautious with regard to judging from the microstructure as to the temperature at which a rail was finished and to be careful in drawing conclusions from the microstructure, alone, in rail-failure investigations. The microstructure appears to be controlled more by the rate of cooling from above the critical point than it does by the temperature at which it was finished. In regular practice rails cool uniformly, since hot rails are going to the bed continually, and therefore the microstructures do not vary as much as they have in this experiment.

Further investigations on this subject are being conducted and the results may be published at some future date.

Rail No.	Minimum Deflection	Maximum Deflection	No of Drops to Break	Total Elongation in 6 In.	Brinell Hardness	Carbon Per Cent.		Tensile Strength	Elastic Limit	Elongation in 2 In.	Contraction of Area
1–A	1.10	4.60	6	0.75	231	0.715	Top	127,000	65,000	18.0	17.02
							Bottom	131,000	68,000	18.5	17.02
1–B	1.10	3.70	5	0.86	234	0.710	Top	127,000	66,000	12.0	17.73
							Bottom	127,000	66,000	13.0	15.93
1–C	1.10	3.60	5	0.90	239	0.700	Top	129,000	66,000	13.0	19.88
							Bottom	132,000	69,000	14.5	19.88
1–D	1.15	5.60	6	0.67	233	0.720	Top	119,000	59,000	16.0	23.38
							Bottom	122,000	62,000	15.0	23.72
1–E	1.15	4.10	6	0.65	241	0.666	Top	122,000	62,000	14.0	20.23
							Bottom	119,000	60,000	15.5	21.63
1–F	1.10	5.15	6	0.78	241	0.622	Top	115,000	60,000	17.0	29.16
							Bottom	116,500	60,000	16.0	30.16
Averages: 1–A, 1–B, 1–C: Fin. Temp. 2,146° F., Shrinkage 6.60 in.; 1–D, 1–E, 1–F: Fin. Temp. 2,020° F., Shrinkage 6.44 in.											
2–A	1.10	5.30	6	0.66	222	0.710	Top	126,500	65,000	11.5	14.11
							Bottom	126,500	65,000	13.0	15.93
2–B	1.15	2.95	4	0.82	227	0.700	Top	124,000	66,000	13.0	15.93
							Bottom	124,000	65,000	13.0	18.09
2–C	1.15	3.00	4	0.92	243	0.730	Top	121,500	63,000	13.5	18.80
							Bottom	123,500	64,000	13.0	18.45
2–D	1.15	3.90	5	0.84	232	0.680	Top	120,000	60,000	14.5	21.29
							Bottom	120,000	59,000	14.5	19.88
2–E	1.20	4.00	5	0.90	223	0.662	Top	119,000	63,000	16.0	24.08
							Bottom	114,000	60,000	18.5	24.08
2–F	1.15	3.95	5	0.84	233	0.650	Top	114,000	58,000	18.0	27.50
							Bottom	112,500	58,000	17.5	26.47

Averages: 2–A, 2–B, 2–C: Fin. Temp. 2,138° F., Shrinkage 6.69 in.; 2–D, 2–E, 2–F: Fin. Temp 2,084° F., Shrinkage 6.63 in.

Rail No.	Minimum Deflection	Maximum Deflection	No. of Drops to Break	Total Elongation in 6 In.	Brinell Hardness	Carbon Per Cent.		Tensile Strength	Elastic Limit	Elongation in 2 In.	Contraction of Area
3–A	1.10	3.80	5	0.69	238	0.704	Top	128,000	65,000	13.5	19.88
							Bottom	128,000	65,000	13.0	20.59
3–B	1.10	3.95	5	0.60	224	0.704	Top	128,000	65,000	12.5	15.56
							Bottom	127,500	66,000	13.0	17.02
3–C	1.10	3.75	5	0.90	234	0.700	Top	123,000	63,000	15.5	24.08
							Bottom	121,000	62,000	16.0	24.08
3–D	1.15	5.00	6	0.79	226	0.690	Top	121,000	63,500	14.0	21.98
							Bottom	112,500	62,000	17.5	25.79
3–E	1.20	4.10	5	0.87	241	0.664	Top	117,500	60,000	15.0	21.63
							Bottom	118,000	60,000	15.0	19.52
3–F	1.15	3.15	4	0.92	234	0.648	Top	113,000	59,000	18.0	26.47
							Bottom	119,000	60,000	15.5	23.03
Averages: 3–A, 3–B, 3–C: Fin. Temp. 2,048° F., Shrinkage 6.64 in.; 3–D, 3–E, 3–F: Fin. Temp. 2,120° F., Shrinkage 6.71 in.											
4–A	1.10	3.80	5	0.72	222	0.720	Top	124,000	63,000	13.0	18.80
							Bottom	128,000	66,000	12.5	18.80
4–B	1.15	3.80	5	0.86	231	0.704	Top	129,000	67,000	18.5	18.80
							Bottom	130,000	67,000	12.5	17.02
4–C	1.10	4.00	5	0.60	235	0.710	Top	125,000	64,000	13.5	21.29
							Bottom	122,000	61,000	15.0	19.52
4–D	1.10	4.05	5	0.68	243	0.686	Top	122,000	64,000	14.0	18.80
							Bottom	118,000	63,000	14.0	17.73
4–E	1.20	5.75	6	0.63	221	0.640	Top	118,500	60,000	15.0	20.59
							Bottom	115,000	60,000	15.5	24.08
4–F	1.25	4.15	5	0.91	227	0.624	Top	113,000	50,000	17.5	27.50
							Bottom	110,000	59,000	18.0	25.79
Averages: 4–A, 4–B, 4–C: Fin. Temp. 2,120° F., Shrinkage 6.77 in.; 4–D, 4–E, 4–F: Fin. Temp. 2,147° F., Shrinkage 6.65 in.											

Rail No.	Minimum Deflection	Maximum Deflection	No. of Drops to Break	Total Elongation in 6 In.	Brinell Hardness	Carbon Per Cent.		Tensile Strength	Elastic Limit	Elongation in 2 In.	Contraction of Area
5–A	1.10	4.90	6	0.80	....	....	Top	126,000	66,000	13.5	17.73
							Bottom	125,000	68,000	7.0	8.90
5–B	1.10	3.80	5	0.83	236	0.700	Top	129,000	67,000	12.0	17.02
							Bottom	129,000	66,000	12.5	18.80
5–C	1.15	3.85	5	0.95	239	0.700	Top	122,000	61,000	14.0	20.59
							Bottom	123,000	63,000	15.0	21.29
5–D	1.15	5.20	6	0.81	234	0.688	Top	121,500	62,000	15.5	19.52
							Bottom	119,000	62,000	14.0	21.98
5–E	1.20	4.10	5	0.98	231	0.654	Top	118,000	60,000	16.0	23.03
							Bottom	116,500	60,000	15.0	23.03
5–F	1.25	4.25	5	0.97	237	....	Top	110,000	56,000	18.0	29.16
							Bottom	111,000	57,000	18.5	27.83
Averages: 5–A, 5–B, 5–C: Fin. Temp. 2,147° F., Shrinkage 6.64 in.; 5–D, 5–E, 5–F: Fin. Temp. 2,048° F., Shrinkage 6.56 in.											
6–A	1.10	3.90	6	0.59	233	0.700	Top	118,000	60,000	16.0	22.33
							Bottom	122,000	61,000	13.5	18.80
6–B	1.10	3.95	5	0.74	237	....	Top	130,000	68,000	12.0	17.02
							Bottom	130,000	68,000	14.0	18.80
6–C	1.15	4.10	5	0.64	234	0.700	Top	122,000	62,500	16.0	24.08
							Bottom	120,000	61,000	15.0	22.33
6–D	1.15	3.00	4	0.78	238	0.710	Top	121,000	63,000	13.5	20.59
							Bottom	120,000	63,000	14.0	17.02
6–E	1.25	3.20	4	0.83	231	0.650	Top	114,000	58,000	15.5	18.80
							Bottom	115,400	59,000	16.0	20.93
6–F	1.20	5.30	7	0.99	233	0.636	Top	113,000	59,000	17.5	26.47
							Bottom	112,500	59,000	18.0	27.50
Averages: 6–A, 6–B, 6–C: Fin. Temp. 2,012° F., Shrinkage 6.37 in.; 6–D, 6–E, 6–F: Fin. Temp. 1,904° F., Shrinkage, 6.37 in.											

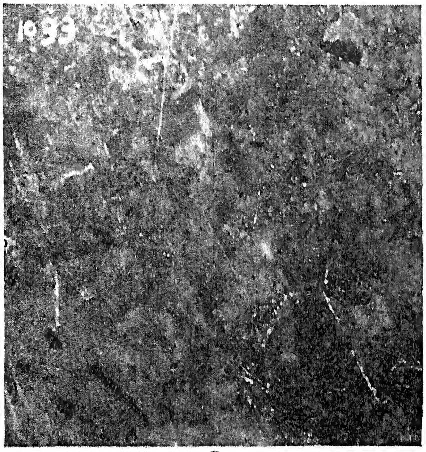
Rail No.	Minimum Deflection	Maximum Deflection	No. of Drops to Break	Total Elongation in 6 In.	Brinell Hardness	Carbon Per Cent.		Tensile Strength	Elastic Limit	Elongation in 2 In.	Contraction of Area
7–A	1.10	3.70	5	0.96	227	0.692	Top	127,000	63,000	13.5	18.09
							Bottom	124,000	63,000	13.5	17.73
7–B	1.10	3.80	5	0.89	235	0.726	Top	124,500	64,000	15.5	22.33
							Bottom	122,230	63,150	15.0	20.78
7–C	1.10	3.75	5	0.98	236	0.700	Top	126,500	65,000	13.5	19.88
							Bottom	123,000	62,000	13.5	20.33
7–D	1.20	4.10	5	0.85	235	0.708	Top	120,000	61,000	16.0	22.33
							Bottom	122,000	62,000	15.5	24.08
7–E	1.20	4.00	5	0.96	239	0.674	Top	109,000	55,000	17.5	29.16
							Bottom	116,000	60,000	16.0	25.45
7–F	1.20	3.15	4	0.92	226	0.640	Top	114,000	60,000	16.0	23.03
							Bottom	113,000	59,000	19.0	30.83
Averages: 7–A, 7–B, 7–C: Fin. Temp. 1,868° F., Shrinkage 6.06 in.; 7–D, 7–E, 7–F: Fin. Temp. 1,850° F., Shrinkage 5.81 in.											
8–A	1.15	3.80	5	0.93	239	0.732	Top	128,000	65,000	14.0	18.80
							Bottom	126,000	65,000	13.0	18.80
8–B	1.10	3.90	5	0.82	226	0.730	Top	124 000	64,000	18.5	22.33
							Bottom	123,000	63,000	19.0	20 59
8–C	1.20	4.90	6	0.88	236	0.738	Top	122,000	60,000	16.0	20.93
							Bottom	124,000	63 000	15.5	20.59
8–D	1.20	3.90	5	1.01	233	0.706	Top	120 000	63,500	15.5	20.59
							Bottom	119,000	61,000	15.0	20.23
8–E	1.20	4.10	5	0.87	235	0.688	Top	116,500	60,000	17.0	25.79
							Bottom	115,000	59,000	17.0	25.79
8–F	1.20	5.30	6	0.88	....	0.650	Top	112,000	55,000	18.5	29.16
							Bottom	110,000	57,000	18.5	29.16
Averages: 8–A, 8–B, 8–C: Fin. Temp. 1,850° F., Shrinkage 5.85 in.; 8–D, 8–E, 8–F: Fin. Temp. 1,841° F., Shrinkage 5.69 in.											

Rail No.	Minimum Deflection	Maximum Deflection	No. of Drops to Break	Total Elongation in 6 In.	Brinell Hardness	Carbon Per Cent.		Tensile Strength	Elastic Limit	Elongation in 2 In.	Contraction of Area
9–A	1.10	3.70	5	0.99	233	0.718	Top	123,000	64,000	12.5	18.80
							Bottom	122,500	62,500	13.0	18.80
9–B	1.15	4.10	6	0.65	233	0.716	Top	126,000	65,000	13.5	18.80
							Bottom	124,000	61,000	15.0	21.19
9–C	1.20	4.05	6	0.62	237	0.708	Top	119,000	60,000	13.5	18.80
							Bottom	116,000	59,000	16.0	22.33
9–D	1.15	5.35	6	0.61	232	0.680	Top	119,000	60,000	17.0	27.16
							Bottom	120,000	60,000	15.5	23.03
9–E	1.20	4.15	5	0.79	221	0.680	Top	118,000	61,500	17.0	24.08
							Bottom	113,000	59,000	17.0	24.76
9–F	1.25	4.25	5	0.90	237	....	Top	112,000	57,000	18.5	30.16
							Bottom	110,000	56,000	19.0	30.83

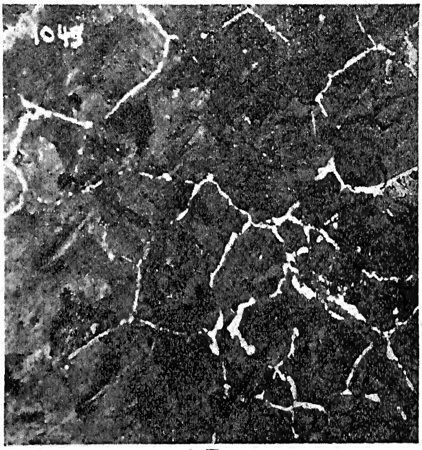
Averages: 9–A, 9–B, 9–C: Fin. Temp. 1,832° F., Shrinkage 5.66 in.; 9–D, 9–E, 9–F: Fin. Temp. 1,652° F., Shrinkage, 5.37 in.

1 A

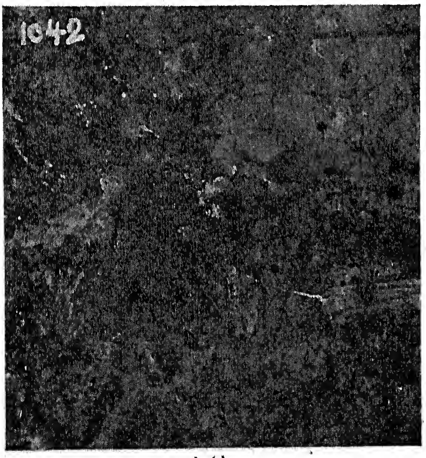
1 D



1 B



1 E



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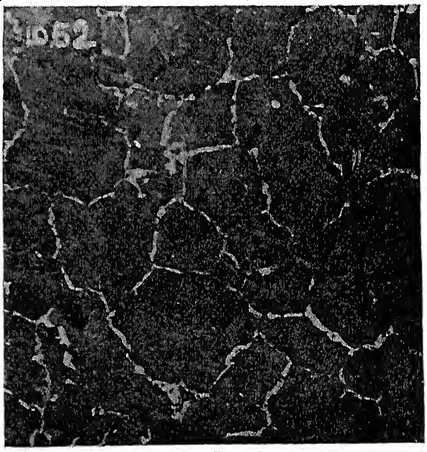
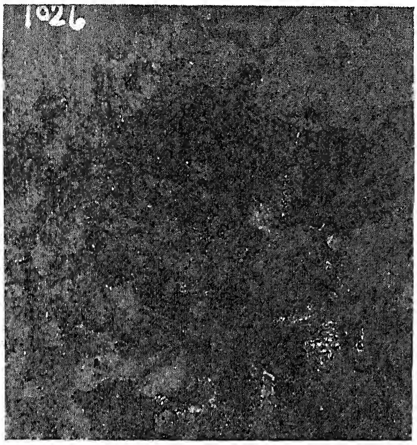
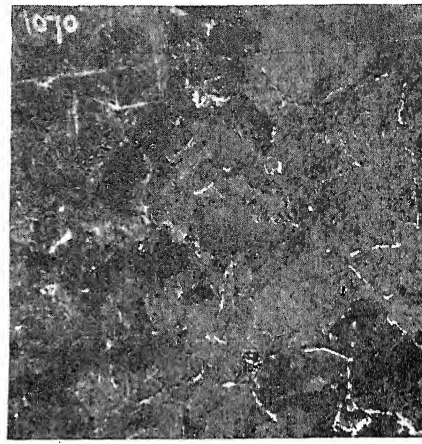


1 F

Fig. 1.—Ingot No. 1. See Tabulated Data, p. 838.

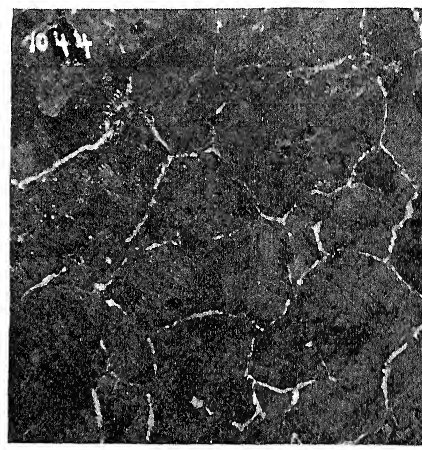


2 A



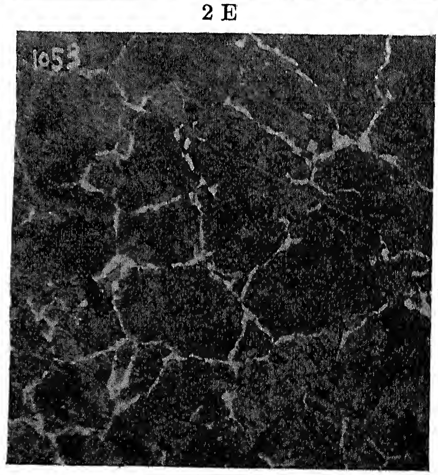
2 D

2 B



2 E

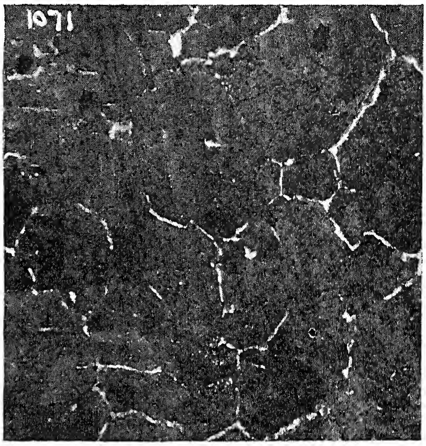
2 C



2 F

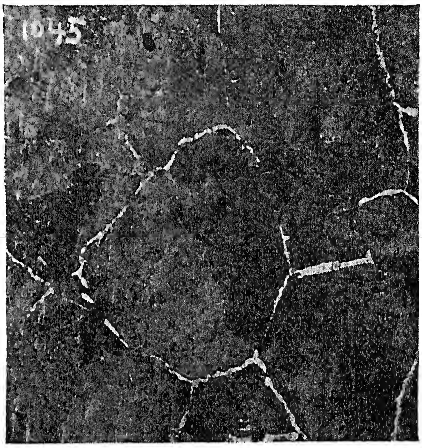
FIG. 2.—INGOT NO. 2. SEE TABULATED DATA, p. 838.

3 A

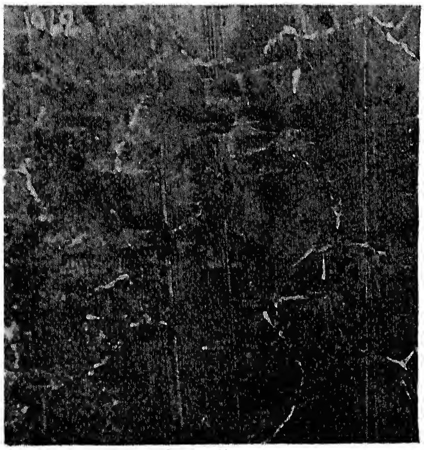


3 B

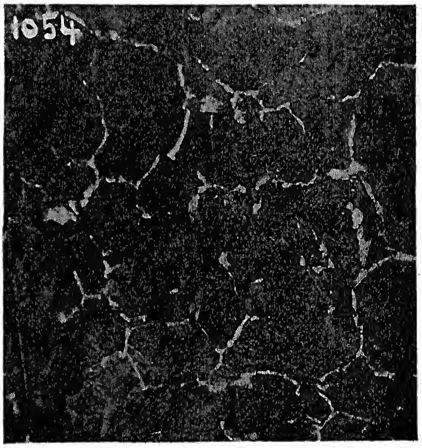
3 B



3 E

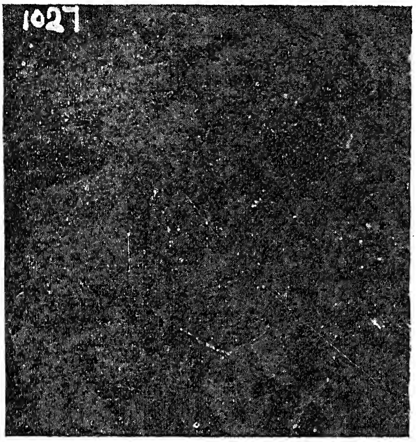


3 C

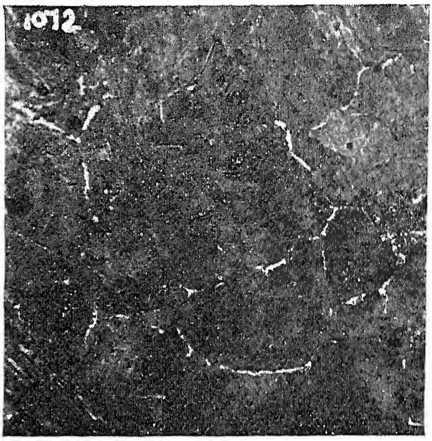


3 F

FIG. 3.—INGOT NO. 3. SEE TABULATED DATA, p. 839.



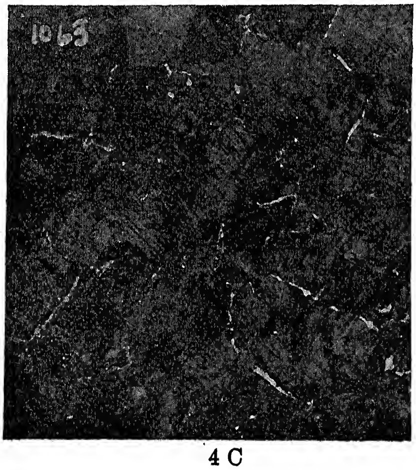
4 A



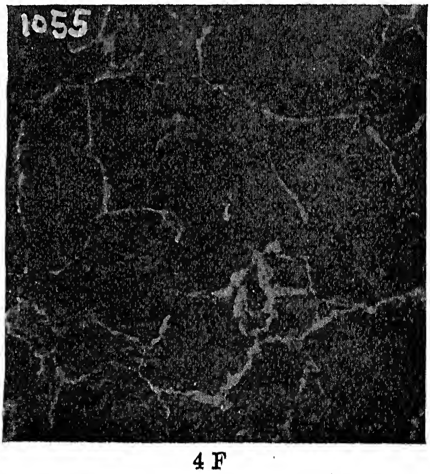
4 D

4 B

4 E

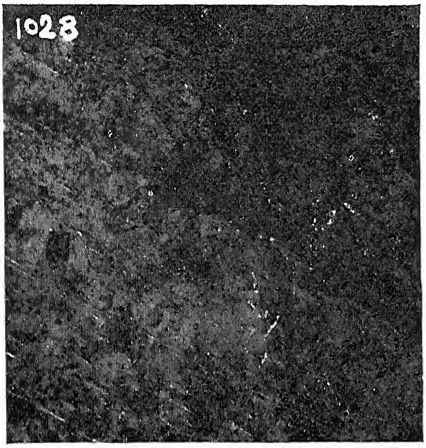


4 C

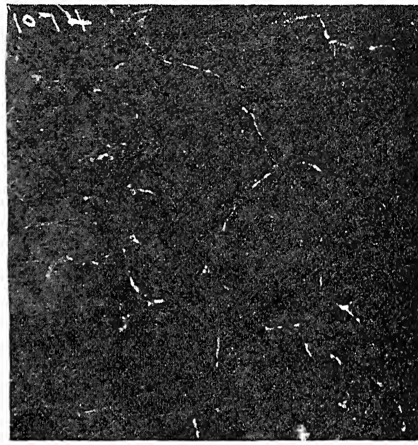


4 F

FIG. 4.—INGOT NO. 4. SEE TABULATED DATA, p. 839.

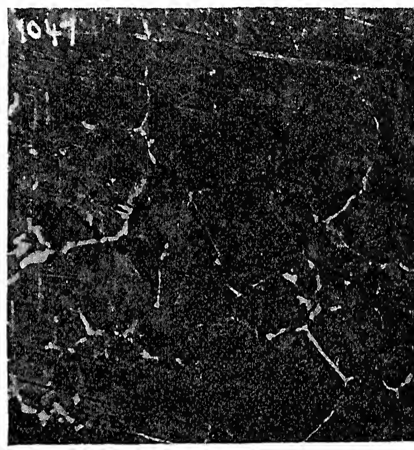


5 A

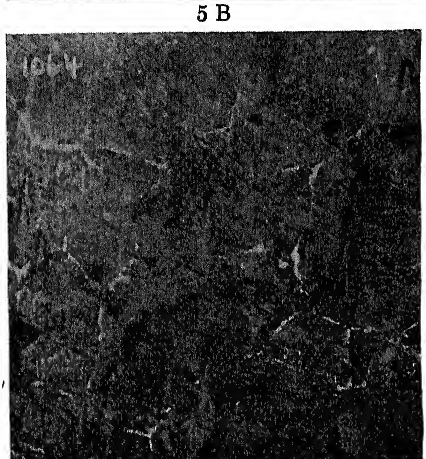


5 D

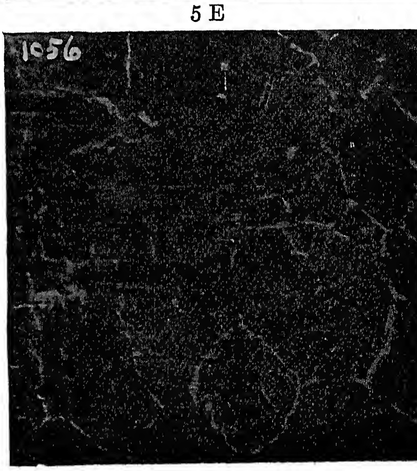
5 B



5 E

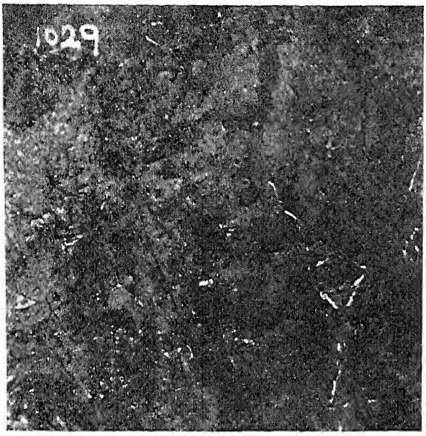


5 C

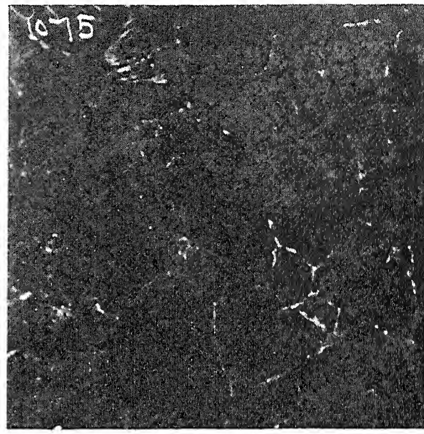


5 F

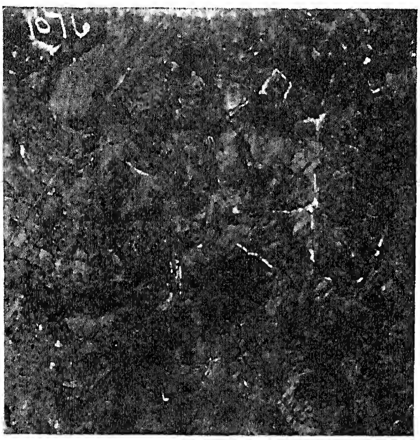
FIG. 5.—INGOT NO. 5. SEE TABULATED DATA, p. 840.



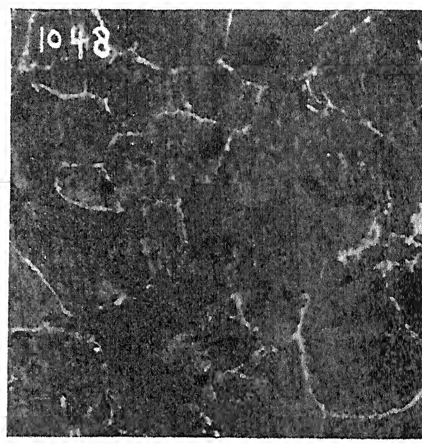
6 A



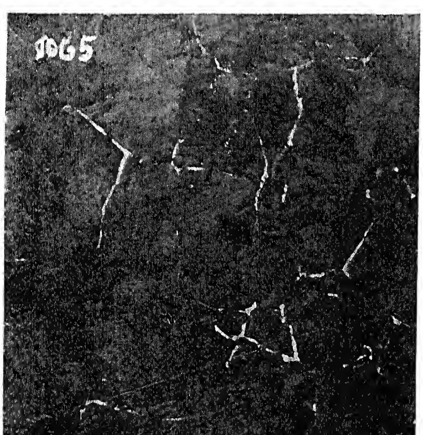
6 D



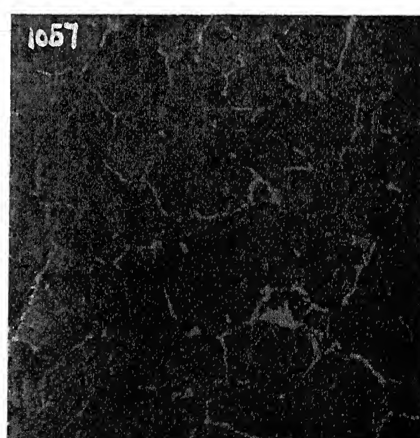
6 B



6 E

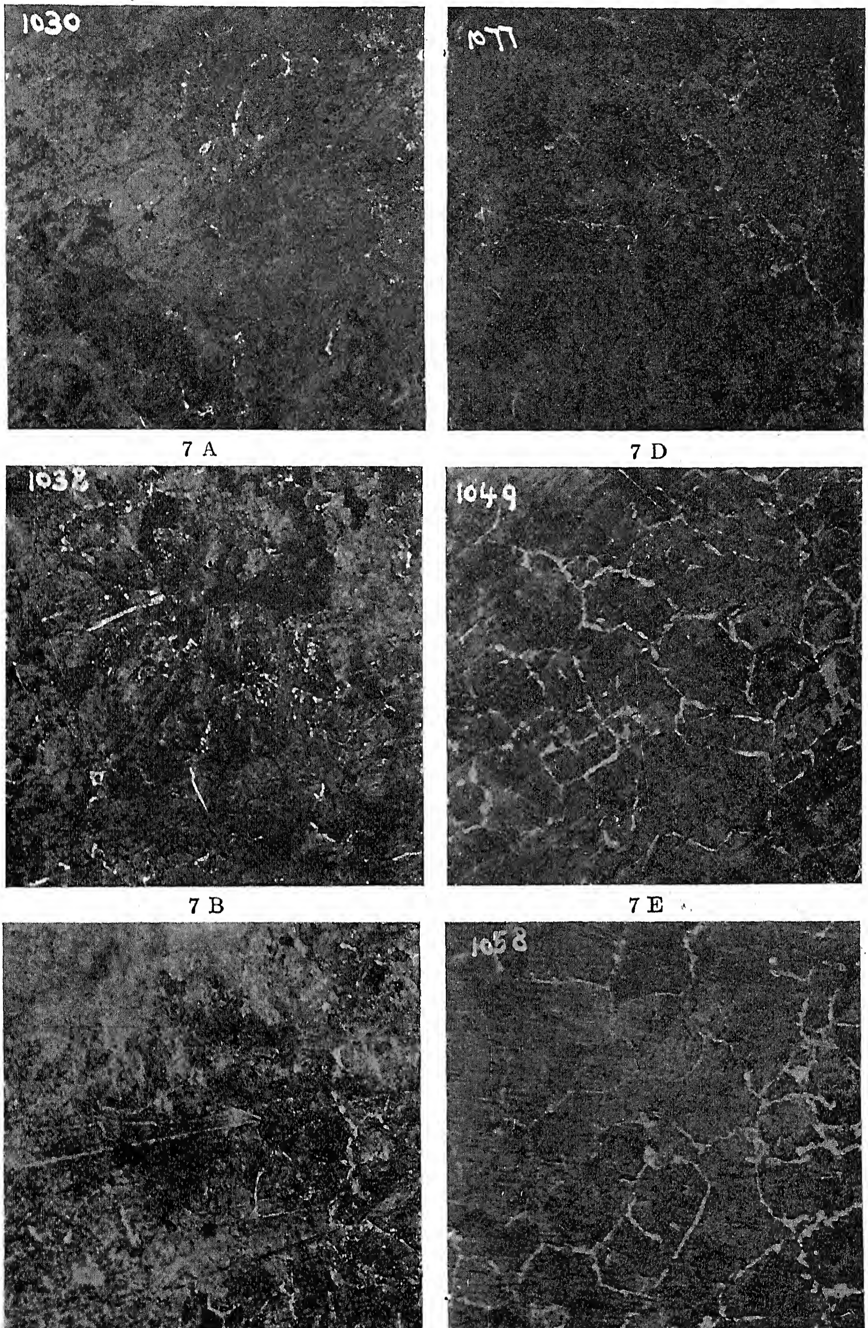


6 C



6 F

FIG. 6.—INGOT NO. 6. SEE TABULATED DATA, p. 840



7 A 7 D

7 B 7 E

7 C 7 F

FIG. 7.—INGOT NO. 7. SEE TABULATED DATA, p. 841.

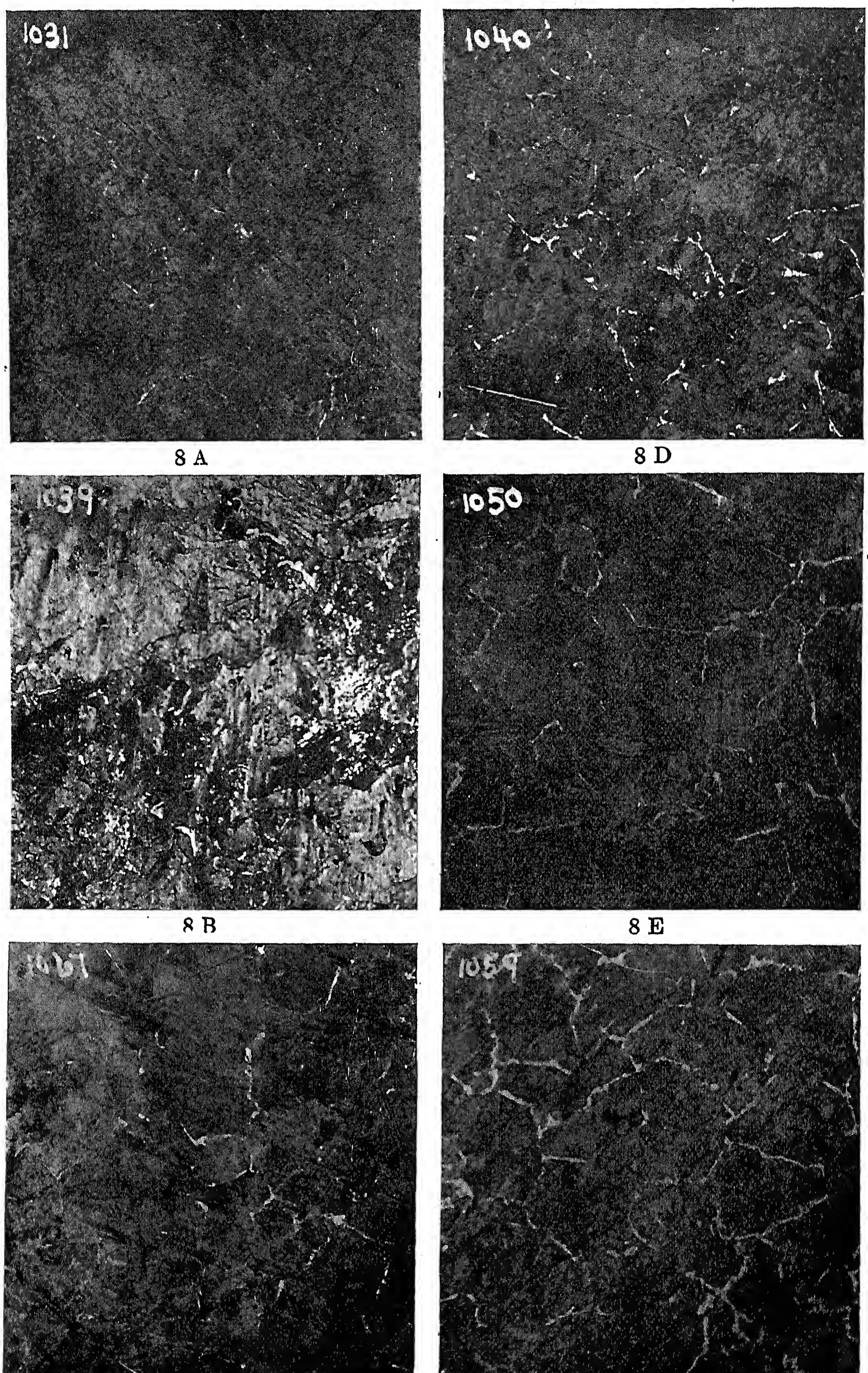
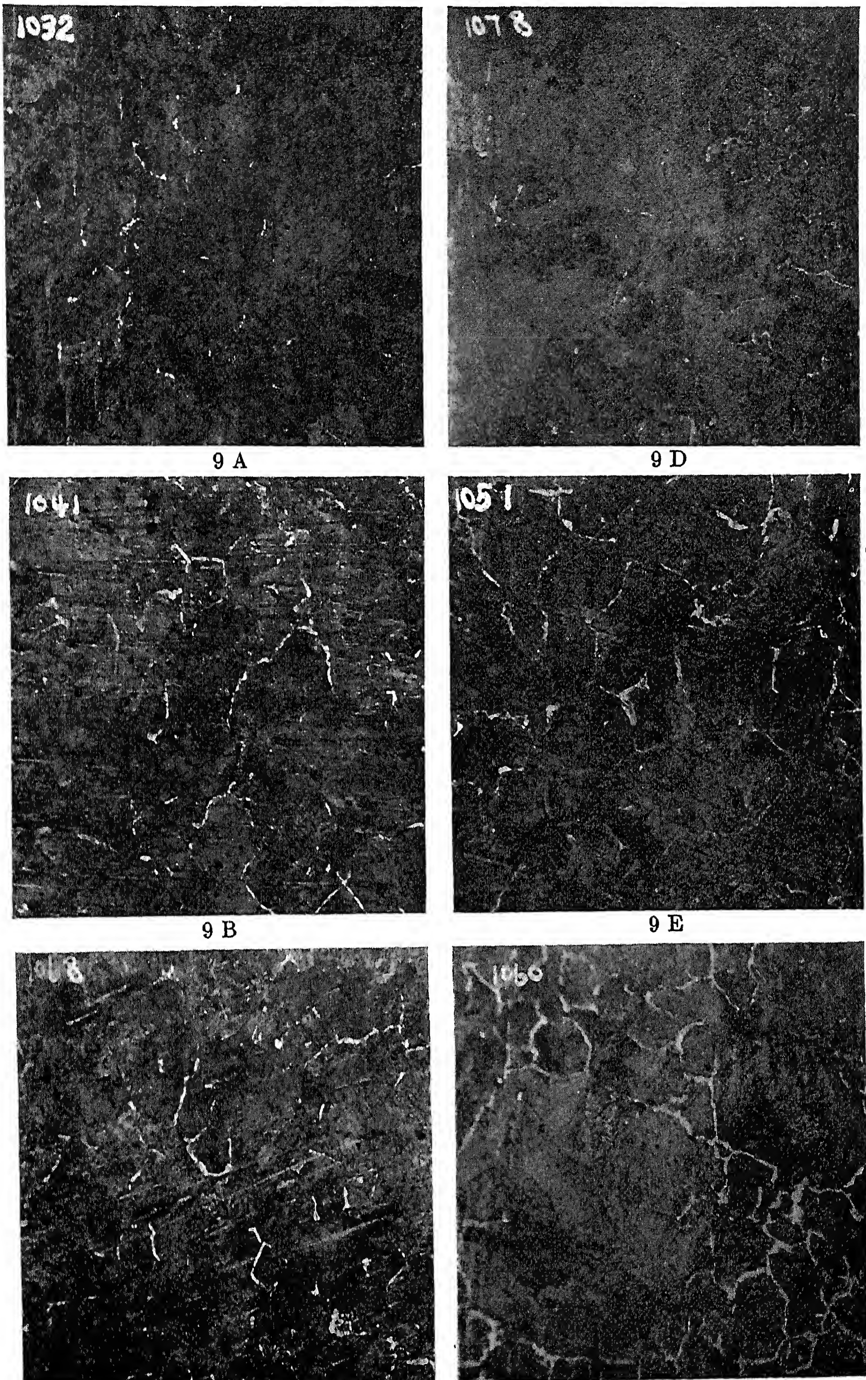


FIG. 8.—INGOT NO. 8. SEE TABULATED DATA, p. 841.



9 A 9 D

9 B 9 E

9 C 9 F

FIG. 9.—INGOT NO. 9. SEE TABULATED DATA, p. 842.

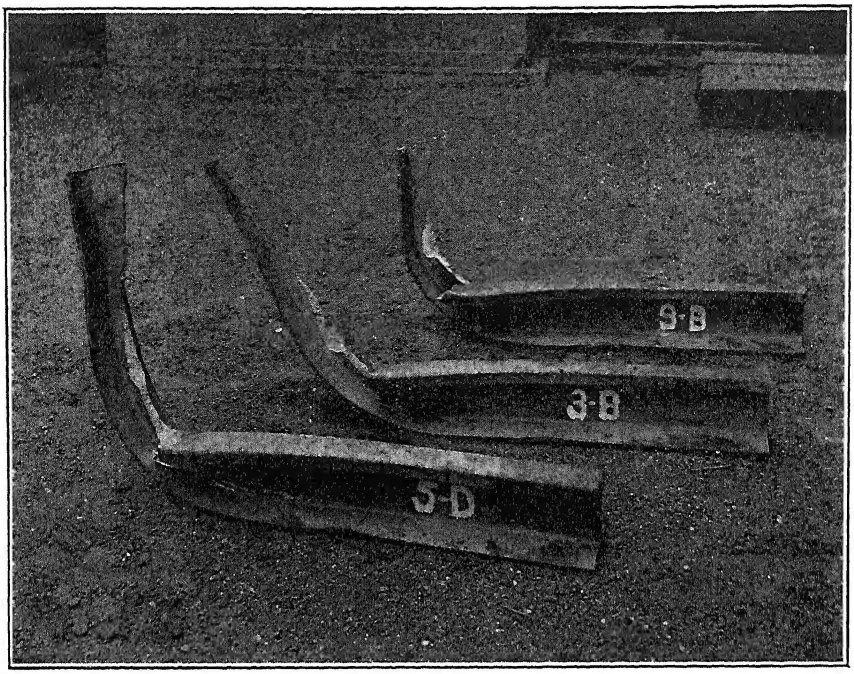


FIG. 10.—SOME OF THE RAILS AFTER BEING SUBJECTED TO DROP TESTS.

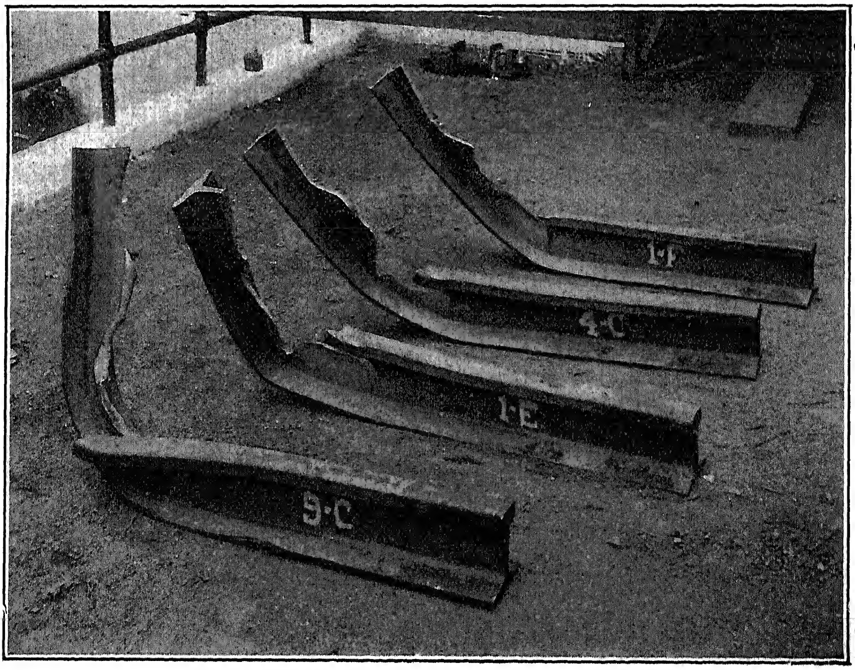


FIG. 11.—SOME OF THE RAILS AFTER BEING SUBJECTED TO DROP TESTS.

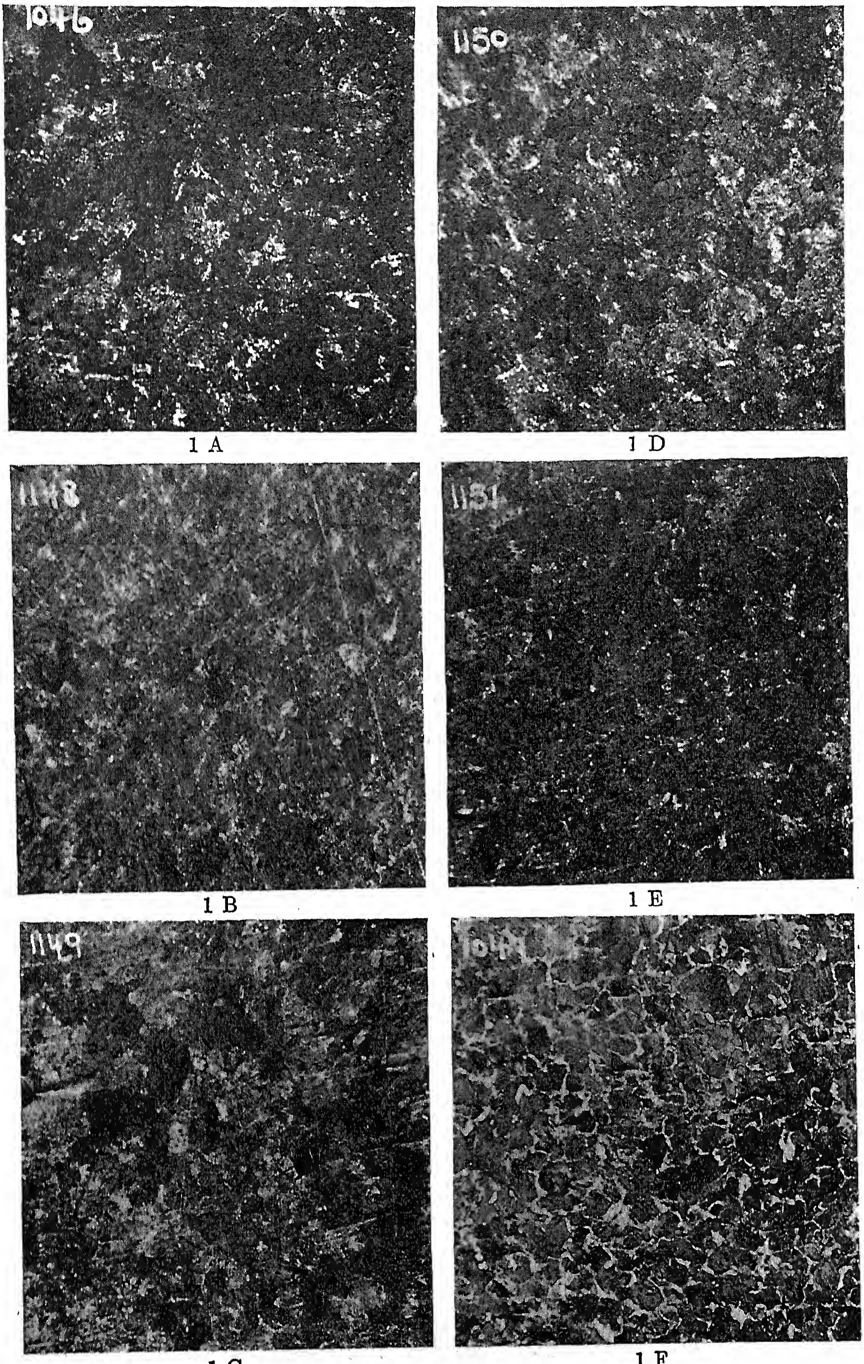
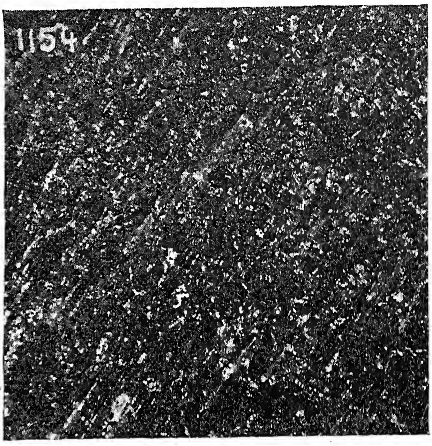
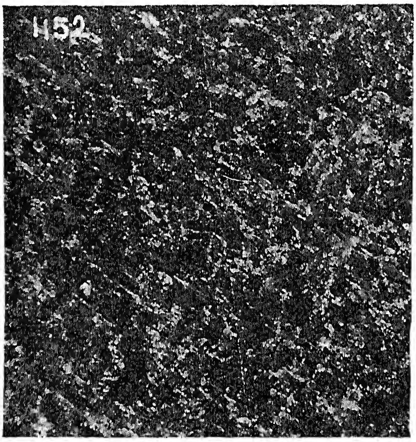


Fig. 12.—Rails 1 A to 1 F. Original Samples Heated to 1,450° F. and Furnace Cooled.

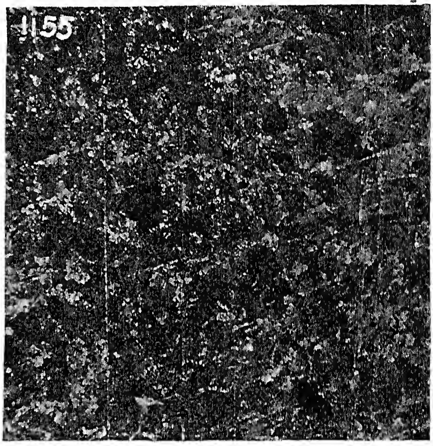
9 A



9 D



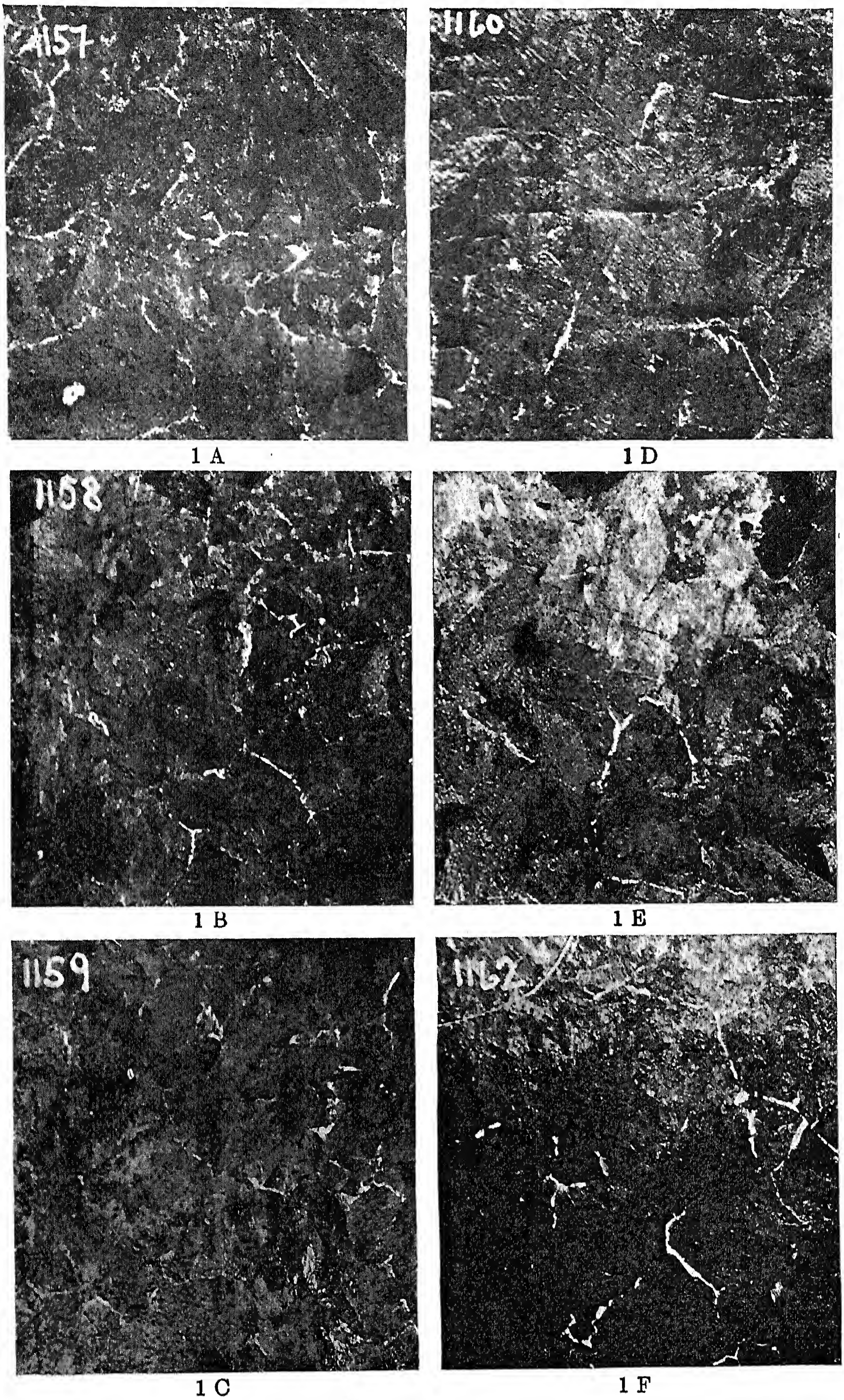
9 B



9 E

9 C

Fig. 13.—Rails 9 A to 9 E. Original Samples Heated to 1,450° F. and Furnace Cooled



1 A 1 D

1 B 1 E

1 C 1 F

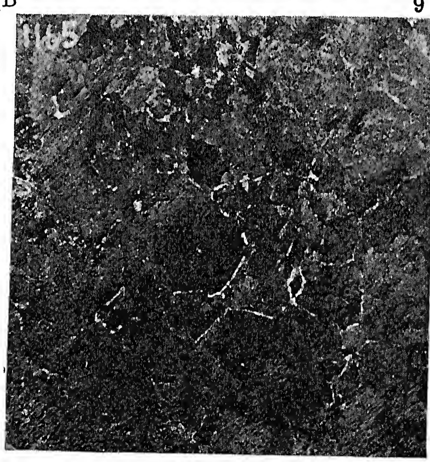
Fig. 14.—Rails 1 A to 1 F. Original Samples Heated to 1,800° F. and Furnace Cooled.

9 A

9 D

9 B

9 E



9 C

Fig. 15.—Rails 9 A to 9 E. Original Samples Heated to 1,800° F. and Furnace Cooled.

Discussion

William R. Webster, Philadelphia, Pa.—I would like to ask Mr. Shimer how the finishing temperature of the H-beams and other beams rolled on the Gray mill at Bethlehem compare with the rolling temperatures of the rails referred to in this paper. Also, how the reductions in the rolling of the beams compare with the reductions in the rolling of the rails. I would also like to know if any drop tests were made on the rails referred to as having been allowed to cool and then annealed; in other words, what results, if any, were obtained from tests on rails after the internal strains of rolling had been removed; and in the matter of the rail blooms, I would like to ask if they are not often heated to a higher temperature than is desirable on account of the thin flanges in order to carry the heat through to the finishing pass.

The tensile strength of the steel in these rails is given as 125,000 lb. per square inch, which is almost as high as tire steel. If some of these blooms were rolled into rounds of about the same section as the head of the rail, would you not get better results from tension tests than those from the head of the rail? In other words, in rolling rounds, whether you would not be better able to control the rolling and finishing temperatures than is possible in the rail, owing to thin section of the rail flanges. Some years ago I had experience with the rolling temperatures of tire steel, running from about 127,000 to 133,000 tensile strength. These tires had the tensile strength and the deflection specified. One of the manufacturers had no trouble whatever, after a little experimenting, in meeting the conditions of the specification. Another manufacturer who made some of these tires came to the office and stated that he had tried tires from three heats of steel and could not meet the specifications and wanted modifications. We examined the report of the chemistry of the steel, results he had achieved, and then asked him to go home and roll some more tires out of the same heats of steel and finish them as cold as he possibly could. He demurred to this, claiming that the tires had been rolled hot in order to meet the requirements. But he did roll the tires from the heats of steel he himself had condemned, and all of the tires met the specifications and were accepted.

I think the effect of finishing temperatures on rails has been merely started, and that all the factors entering into the problem will have to be considered before we can come to any definite conclusions. Will we in the end arrive at a better section of rail that will allow a better control of the reductions in rolling and finishing temperatures? Under present conditions we cannot give the steel fair treatment in rail sections with the wide, thin flanges.

Henry M. Howe, Bedford Hills, N. Y.—One thought in this connection is, perhaps, that in view of such a high manganese content we need not expect so much influence from finishing temperature as we do in such material as Mr. Webster speaks of, because of the sluggishness of the metal in the presence of so much manganese. Very likely the question of finishing temperature may be less important in a 1 per cent. manganese material than in the material which Mr. Webster has been using, which is probably very much more sensitive to the finishing temperature.

* The probable key to the discrepancy is the influence of time. There is no doubt that we formerly attached too much importance to temperature and too little to time. Later investigations show that time is quite as important as temperature.

The opportunity for coarsening exists between the moment when the rail last undergoes sufficient deformation to break up the austenite grains and causes them to form anew and the moment when it finally cools below Ar3. The actual finishing temperature, as measured by the temperature at which the rail leaves the finishing pass, may be misleading, because the reduction in that pass and indeed in the leading pass and the one preceding may be insufficient to break up the existing austenite grains effectively.

Beyond this the rate of cooling of so thin a section of rail is likely to be so rapid as to cut down materially the opportunity for growth of the austenite grains. When a specimen is heated up to a given temperature, say 1,000°, held there and allowed to cool down slowly, the opportunity for grain growth includes first the period between leaving Ac3 and reaching 1,000°, second the stay at 1,000°, third the time in cooling from 1,000° to Ar3.

What with hastening the cooling by water played on the rail, and with the rapid natural rate because of the thin section of the rail, the opportunity for coarsening is relatively short.

These conditions may explain why the influence of finishing temperature is less marked than might have been expected.

A. A. Stevenson, Philadelphia, Pa.—The manganese content of the tires referred to by Mr. Webster was 0.70 to 0.80, sometimes 0.90.

G. K. Burgess, Washington, D. C.—I am very glad to see that Mr. Shimer has been able to carry out these experiments and also has had the opportunity to publish them. One or two matters occur to me which suggest a possibility of the reason for some of the indeterminateness of the results. As I understand it, the blooms were taken from a reheating furnace and allowed to cool. I think there is some question in

* Communication to the Secretary. Received Mar. 24, 1915.

that operation as to the uniformity of the temperature throughout the bloom. Although Mr. Shimer has apparently, when he measured the temperatures on the surfaces of the finished rails, the characteristic condition which gives him an indication of the temperature of rolling, I think if it had been possible to control the temperature of his reheating furnace so that the bloom was unquestionably reheated throughout to the same temperature, then a source of indeterminateness would have been removed. However, there was, of course, some variation in temperature at least across a section of the rail, but in so far as these experiments prove anything, they prove that there is no very considerable effect due to finishing temperature on the properties of the rail.

Just a word regarding the microstructure. There is one inconsistency, at least it appears to me such, which I find difficult to understand. As it appears, in Mr. Shimer's explanation of the placing of the rails on the bed, he accounts for the variation in structure due to the difference in cooling, but his description of placing them on the beds, it seems to me, would make the *B* and the *E* rails receive identical cooling, yet in all cases the *B* and *E* rails show a very considerable difference in structure. Also, regarding any inconsistency between the shrinkage and finishing temperature, of course, when both are correctly taken, there should be no discrepancy there.

I think it is also of interest to mention the fact that within the last day or two a report was made by Mr. Wickhorst, of the American Railway Engineering Association, in which, so far as that report is conclusive, there is some slight evidence that a lower finishing temperature is advantageous in the properties of the rail. I think, however, that Mr. Wickhorst is not very emphatic on that statement.

Mr. Shimer's paper is a useful contribution to the subject, and it is hoped he will be able to execute the further experiments contemplated.

Samuel L. Hoyt,* Minneapolis, Minn. (communication to the Secretary†).—The general conclusion to be drawn from this paper seems to be that the mechanical testing of rails, as now practiced at the steel plant, fails to develop the true quality of steel rails.

The author states in the first sentence of the summary that "No appreciable difference in grain was found to indicate that the size or structure was governed by a difference in finishing temperatures." Such a statement would bear more weight if accompanied by a table giving the cell size of each specimen, at least for those specimens in which the free ferrite has separated out as a network.

The rails can be conveniently divided into two groups: (*a*) those with no free ferrite and (*b*) those with free ferrite. The rails under (*a*) consist chiefly of sorbite, so that the grain size is too small to be determined

* Assistant Professor of Metallography, University of Minnesota.

† Received Mar. 12, 1915.

microscopically. The properties, so far as they are dependent on the grain size, would hardly be expected to vary appreciably. Any variation in the properties of these rails would more probably be due to difference in chemical composition, etc. Under (*b*) the grain size is likewise too small to be conveniently determined. There are, however, two important points in connection with the microstructure of these rails: the size of the cells, and the amount and distribution of the free ferrite contained in the network. These factors, under the conditions of the experiment, should vary with the finishing temperature.

For the sake of illustrating this point, I have compiled the following table, which gives the finishing temperatures of the *E* and *F* rails, tabulated consecutively, as well as the total number of cells contained within the microphotographs as closely as could be estimated, also tabulated consecutively. This table is hardly to be looked upon as being strictly accurate, but the results are so promising that the cell size, as measured by some one of the usual methods, might be advantageously added for the sake of comparison. Such a table would probably be helpful in connection with the finishing temperatures.

Cell Size of the E and F Rails vs. Finishing Temperature

No.	Finishing temperature, °F.	No.	Cells	No.	Cells
4	2,147	4 *E*	12	5 *F*	23
3	2,120	3 *E*	13	4 *F*	24
2	2,080	5 *E*	21	3 *F*	28
5	2,048	2 *E*	28	2 *F*	30
1	2,020	6 *E*	32	1 *F*	41
6	1,904	1 *E*	37	6 *F*	45
7	1,850	8 *E*	40	7 *F*	46
8	1,841	7 *E*	45	8 *F*	48
9	1,652	9 *E*	50	9 *F*	52

The author refers to the drop test as "the important test of the quality of the rail." This view is being somewhat discredited of late, on account of lack of discrimination. The mill tests are always on fresh steel and of very short duration. The service test of the rail, while in the track, is of comparatively long duration and on steel which is continually aging. A defect, sufficiently pronounced to cause a "rail failure," need not necessarily cause the rail to give abnormal results either in the testing machine or under the drop test, because the mechanism bringing about the failure in the two cases is different.

That the ordinary mill tests are more or less crude is substantiated by the conclusion of Burgess and Hadfield in their paper on the Comparison Tests of Rail Ingots that "The usual physical and chemical tests do not appear to give adequate measure of the quality of rails." The irregulari-

ties given in the tables of the present paper are further evidence of the crudities of these determinations of the quality of steel rails. Thus in the second half of the table on p. 835, rails *D*, *E*, and *F*, No. 4, have a lower carbon content than No. 9 of the same group, yet No. 4 is stronger and has a lower elongation and reduction of area than No. 9. (The value of 30.00 for No. 9 is evidently incorrect and should be 26.66.) This shows there must be factors other than small variations in the carbon content which have an influence on these results, and which are not brought out in the tests.

The higher finishing temperature and lower carbon content of No. 4 as compared to No. 9 should give results exactly the opposite of these. The above could be overlooked if the absolute value of the material alone were desired, but when relative values are needed, all the variables save one (finishing temperature) should be kept constant. It is apparent that, in ordinary plant practice, even though carefully conducted, there are too many variable factors, the influences of which are sufficient to obscure the results sought. Even at that it is rather difficult to understand the author's meaning in "all the tensile and drop tests recorded are good, and *practically identical*."[1] For example, the elastic limits (yield points?) vary by about 16 per cent. in the tables given, etc.

The author's statement to the effect that the strains set up during the reduction of the ingot have an effect on the properties of the finished section is at variance with the generally accepted theory that (with excessive reduction excepted) it is the grain and cell size, produced during the hot rolling and on cooling, that influences the properties. The strains set up during "hot rolling," if one chooses to consider it this way, are relieved immediately (or nearly so) during the recrystallization which accompanies the reduction in grain (crystal) size.

[1] The italics are mine.

Sound Steel Ingots and Rails*

BY GEORGE K. BURGESS,† WASHINGTON, D. C., AND SIR ROBERT A. HADFIELD,‡ LONDON, ENGLAND

(New York Meeting, February, 1915)

1. *Introduction.*—THE methods of production of sound steel ingots have been described in several papers read recently before this Institute. It was thought by Director Stratton, of the U. S. Bureau of Standards, and ourselves to be of sufficient interest and importance, as a further contribution to this subject, to compare the properties of rails rolled in one mill according to American practice, from several types of ingot of substantially the same chemical composition but cast by several processes; and to compare in considerable detail the properties of an ingot manufactured by the Hadfield special-feeding process and one of the usual type for rolling into rails.

We are greatly indebted to F. W. Wood, President of the Maryland Steel Co., who made this investigation possible by kindly placing his rail mill at our disposal; and he and his associates at Sparrows Point did everything possible to make the experiment a success, including the execution of several of the tests and analyses.

2. *Classification and Manufacture of Ingots.*—Of the ingots used in this investigation, eight were made in Sheffield and furnished by Sir Robert Hadfield, and one was made and furnished by an American steel company,[1] as a comparison ingot, and was supposed to represent the usual type of ingot from which rails are made.

The characteristics of the ingots are given in Table I. Ingots 1, 2, 3, and 4 were cast large end up and fed by the Hadfield method in the usual manner with charcoal and blast continued until the molten steel has set on the top of the head, say varying from 20 to 40 min. To ingots 1, 2, and 4 was added 0.1 per cent. aluminum, and to the nickel-chrome ingot, No. 3, 0.125 per cent. For sake of comparison, there were also included two ingots, Nos. 8 and 9, cast with the small end up and fed by

* A complete account of this investigation, giving all data in detail, will be published later as a Technologic Paper of the U. S. Bureau of Standards.

† U. S. Bureau of Standards. ‡ Honorary Member.

[1] This ingot is not to be assigned to any particular manufacturer, as it is one of several from different sources, supplied without special selection as typical of the usual output of ingots.

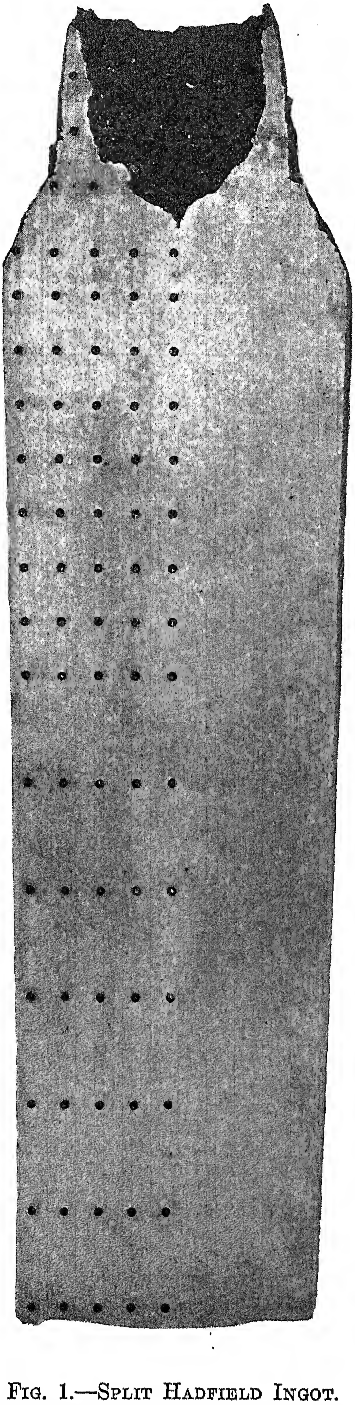
the Hadfield process; three ingots not fed, two of piping steel, Nos. 6 and 10, and one of rising or "unsound" steel, No. 7. The Hadfield ingot No. 1 and the American comparison ingot, No. 10, were cut up for examination as to segregation, blowholes, and piping, while the others were rolled into rails. It was expected that ingots Nos. 2, 3, and 4 would give rails of a uniform quality throughout nearly the whole length of the ingot, and that No. 1, which was selected by chance from the specially

TABLE I.—*Classification of Ingots*

No.	Size, Inches	Weight, Pounds	Method of Casting	Type of Steel	Chemical Analysis				
					C	Mn	S	P	Si
1	18	5,964	Hadfield system, large end up.	Ordinary rail steel.	0.59	0.97	0.0410	0.0310	0.21
2	18	6,006	Hadfield system, large end up.	Ordinary rail steel.	0.55	0.96	0.0430	0.0290	0.12
3	18	5,900	Hadfield system, large end up.	Nickel-chromium steel.	0.25	0.38	Ni 3.700	Cr 1.470	0.19
4	18	5,850	Hadfield system, large end up.	Ordinary rail steel.	0.56	0.92	S 0.040	P 0.035	0.19
6	18	5,200	Ordinary manner, small end up.	Ordinary rail steel which pipes, not fed.	0.56	0.94	0.060	0.051	0.20
7	18	5,100	Ordinary manner, small end up.	Ordinary rail steel of rising or boiling nature, not fed.	0.57	0.95	0.080	0.045	0.20
8	18	5,700	Hadfield system, small end up.	Ordinary rail steel.	0.58	0.96	0.070	0.051	0.23
9	18	5,700	Hadfield system, small end up.	Ordinary rail steel.	0.59	0.96	0.054	0.040	0.20
10	19	....	The American ingot, cast in ordinary manner, small end up.	Ordinary rail steel.	0.46	0.94	0.050	0.090	....

fed ingots cast large end up, would show a uniform, sound structure throughout. Previous experience has shown that ingots of the type of Nos. 8 and 9, cast by the Hadfield process with small end up, would be expected to be less satisfactory than the Hadfield type represented by Nos. 2, 3, and 4, cast with large end up. The former sometimes show a tendency to develop a pipe at the lower end. Ingot No. 6 would be expected to show piping and segregation, and No. 7 segregation and unsoundness; for both of these ingots a much greater waste would be expected than for the specially fed ingots.

3. *Examination of Ingots.*—At the Pittsburgh Laboratory of the Bureau of Standards two of the ingots were sawed in halves—Hadfield No. 1, of the specially fed type cast large end up, and the comparison ingot, No. 10, from an American mill. Figs. 1 and 2 are photographs of the two ingots showing the location of drillings and a comparison of the interiors of the two ingots. The Hadfield ingot was also smoothed off in order to take a sulphur print, shown in Fig. 3. In Figs. 4, 5, and 6,



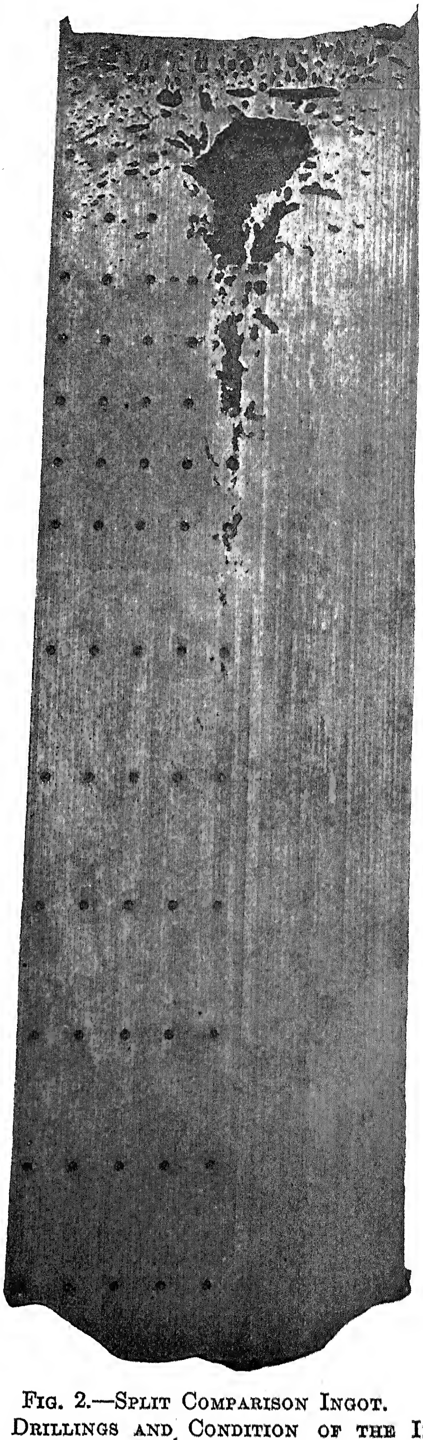


FIG. 1.—SPLIT HADFIELD INGOT. FIG. 2.—SPLIT COMPARISON INGOT.

FIGS. 1 AND 2.—SHOWING LOCATION OF DRILLINGS AND CONDITION OF THE INTERIOR OF THE INGOT.

TABLE II.—*Rolling and Inspection of Ingots and Rails*

Number of ingot	3	2	4	6	7	8	9
Taken from pit at	8 hr. 9 min.	8 hr. 40 min.	8 hr. 55 min.	9 hr. 11 min.	9 hr. 27 min.	9 hr. 43 min.	10 hr. 45 min.
Time in blooming mill	1 min. 40 sec.	1 min. 40 sec.	1 min. 42 sec.	1 min. 30 sec.	1 min. 40 sec.	1 min. 30 sec.	1 min. 40 sec.
Average temperature of ingot in blooming mill, deg. C.	1,100°	1,115°	1,110°	1,110°	1,135°	1,150°	1,125°
Time blooming mill to hot saw.	3 min. 50 sec.	4 min. 10 sec.	3 min. 23 sec.	4 min. 5 sec.	3 min. 20 sec.	3 min. 15 sec.	3 min. 22 sec.
Average temperature first rail bar at hot saw.	950°	960°	970°	970°	970°	980°	980°
Average temperature second rail bar at hot saw.	945°	900°[a]	950°	945°	955°	970°	965°
Top bloom butt, per cent. of ingot.	4.2[b]	5.6[b]	7.8	12.9	5.0	5.1	6.3
Total bloom butts, per cent.	7.2	7.4	.9.7	14.7	8.0	6.8	8.3
Oxidation loss, per cent.[d]	1.3	2.7	1.9	2.9	3.5	3.5	2.5
Percentage of ingot available for rails.	91.5	89.9	88.4	82.4[c]	88.5	89.7	90.2
Rails: Percentage of firsts.	77.2	77.1	52.5	9.5	54.3	51.9	28.5
Percentage of seconds[f]	22.8	22.9	47.5	90.5	27.4	48.1	23.8
Percentage of scrap[f]	0	0	0	0	18.3	0	47.7[e]
Type of ingot	Special	feed, large	end up	Ordinary piping	Ordinary rising	Special feed, s	mall end up

[a] Was held back before coming to saw.
[b] Has been corrected as explained in §4.
[c] Contains pipe, however, extending into *C* rail.
[d] It is to be kept in mind that all are reheated ingots.
[e] Two rails, *C* and *D*, from middle of ingot were scrapped for mechanical defects.
[f] This classification includes, of course, defects due to mill practice.

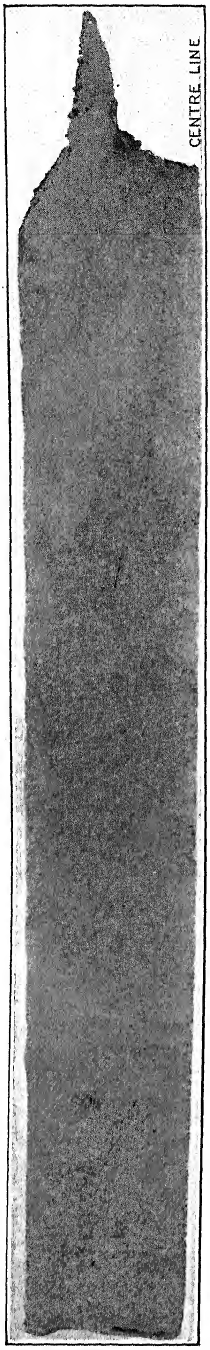


Fig. 3.—Photograph of Sulphur Print of Half-Section of Hadfield Ingot.

respectively, are shown the segregation of carbon, phosphorus, and sulphur for each of the ingots. A similar survey was also made for manganese, with no considerable segregation of this element in either ingot. It is evident that there is freedom from appreciable segregation over 95 per cent. or more of the Hadfield ingot, which is also entirely free from piping or blowholes. The comparison ingot, if rolled into rails, should have about a 50 per cent. discard.

4. *Manufacture of Rails.*—The ingots were charged into the soaking pits, which had been allowed to cool considerably, on Sunday evening, and rolled Monday, Apr. 13, 1914, at 8 A. M., into 100-lb. P. S. section rails, the seven ingots being interpolated among other ingots. In the Maryland mill the rolling from ingot to finished rail is a continuous process without reheating of the blooms. In Table II are shown some of the characteristics of the rolling. Apparently through a misunderstanding of instructions and unfamiliarity with this type of ingot, the top bloom crops of the first two rolled, specially fed Hadfield ingots Nos. 2 and 3, were excessive, at least double what was necessary to carry the blooms through the rail mill. The other Hadfield and comparison ingots were sheared as instructed. The *A* rail was made the short rail, usually 6 to 10 ft. long, to be examined as to soundness. The drop-test specimen was cut next the bottom of this short *A* rail, a 13-in. specimen for tensile tests from the head of the *C* rail, and 3-in. pieces for chemical, metallographic, and hardness tests from the heads of the *A*, *B*, and *C* rails.

5. *Tests of Rails.*—In Table III are shown the Brinell hardness tests taken at the top of the *A* and *C* rails for positions shown in Fig. 7. It is of interest to note that the tests, with the exception of those on rails with a pipe or seam (from ingots Nos. 6, 8, 9), show a high value of hardness numeral for position 5 in the *A* rail. Otherwise the hardness is quite uniform over the sections, and from one rail to another, except, of course, for the nickel-chromium, No. 3.

The results obtained by the Maryland Steel Co. for the drop tests are shown in Table IV; of inspec-

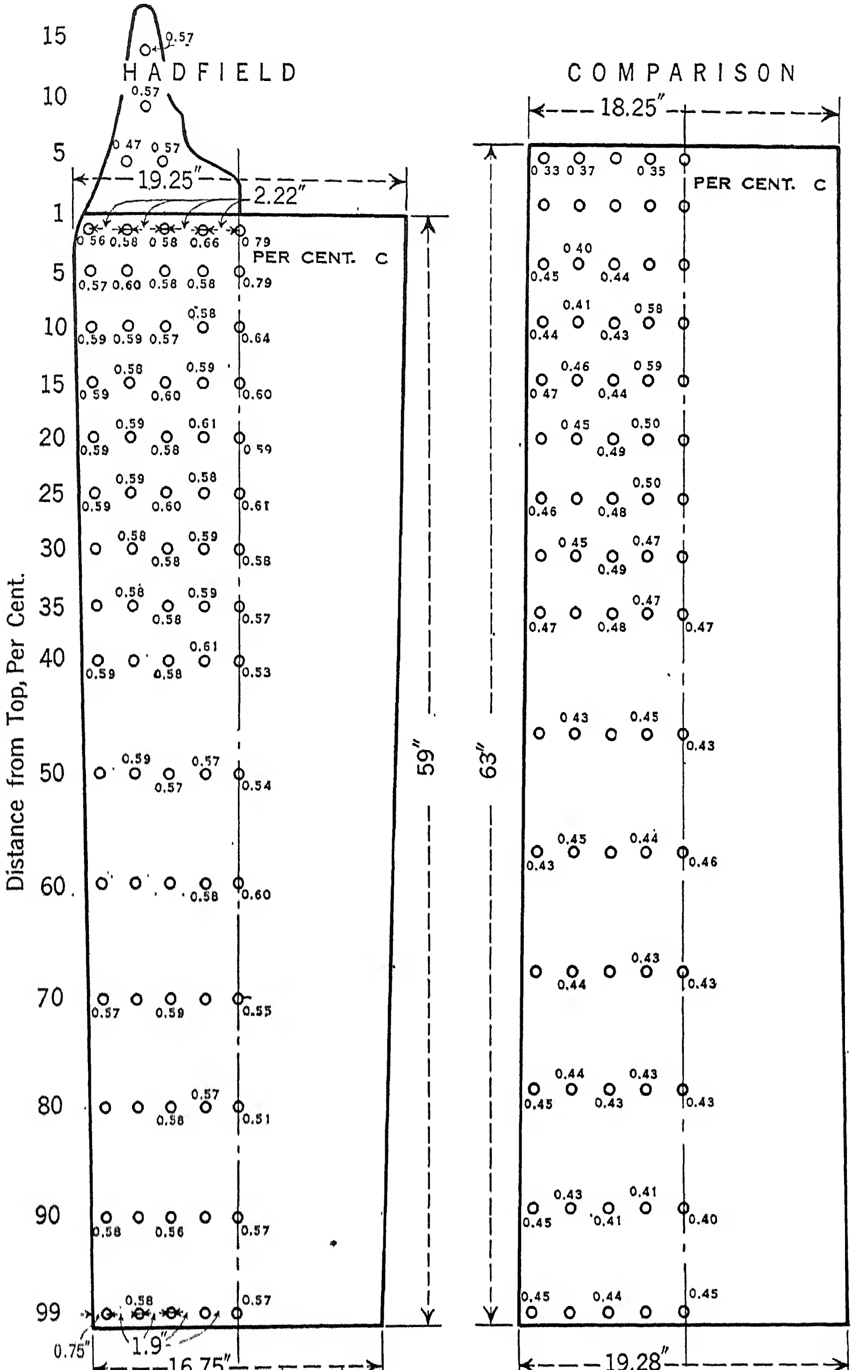


FIG. 4.—CARBON SEGREGATION.

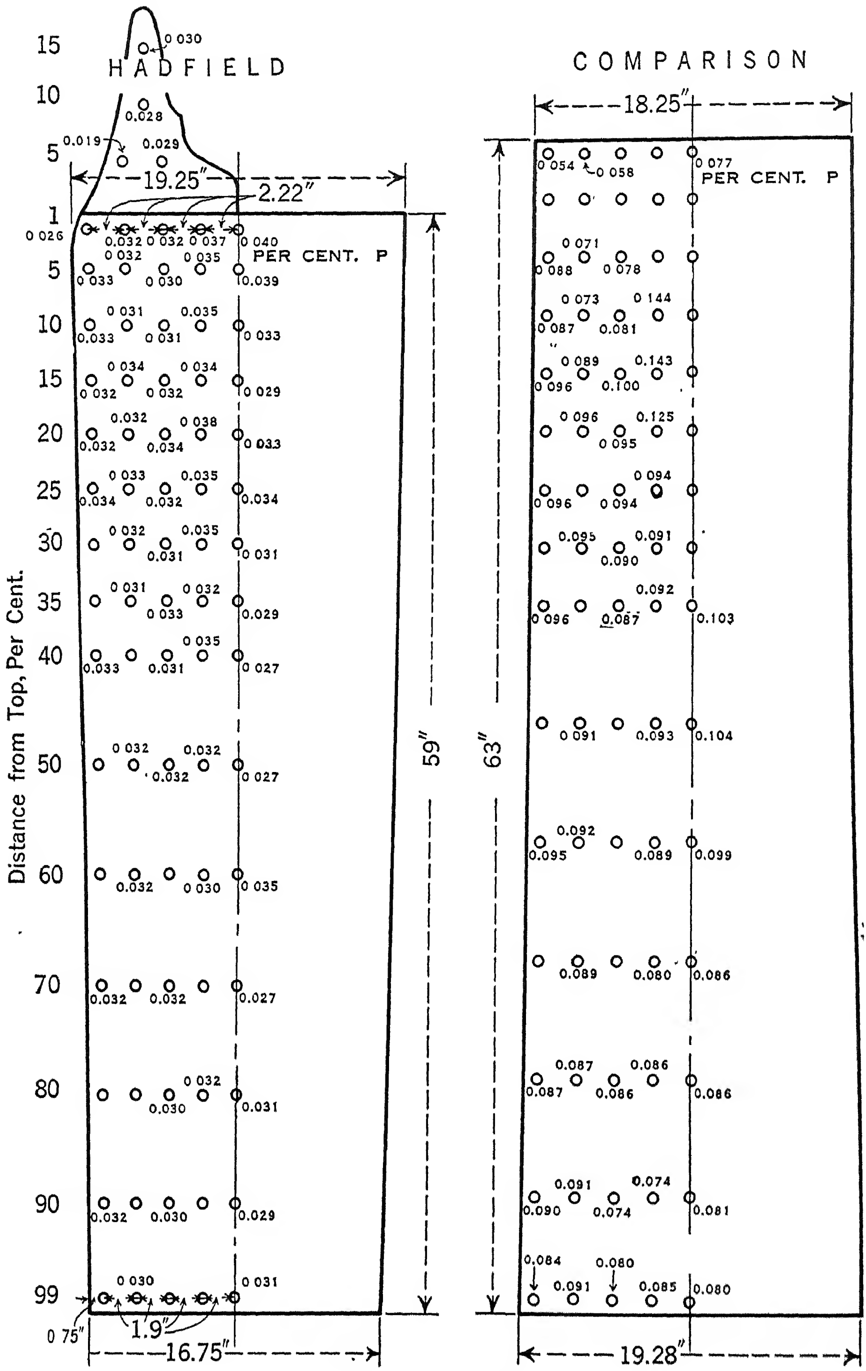


FIG. 5 —PHOSPHORUS SEGREGATION.

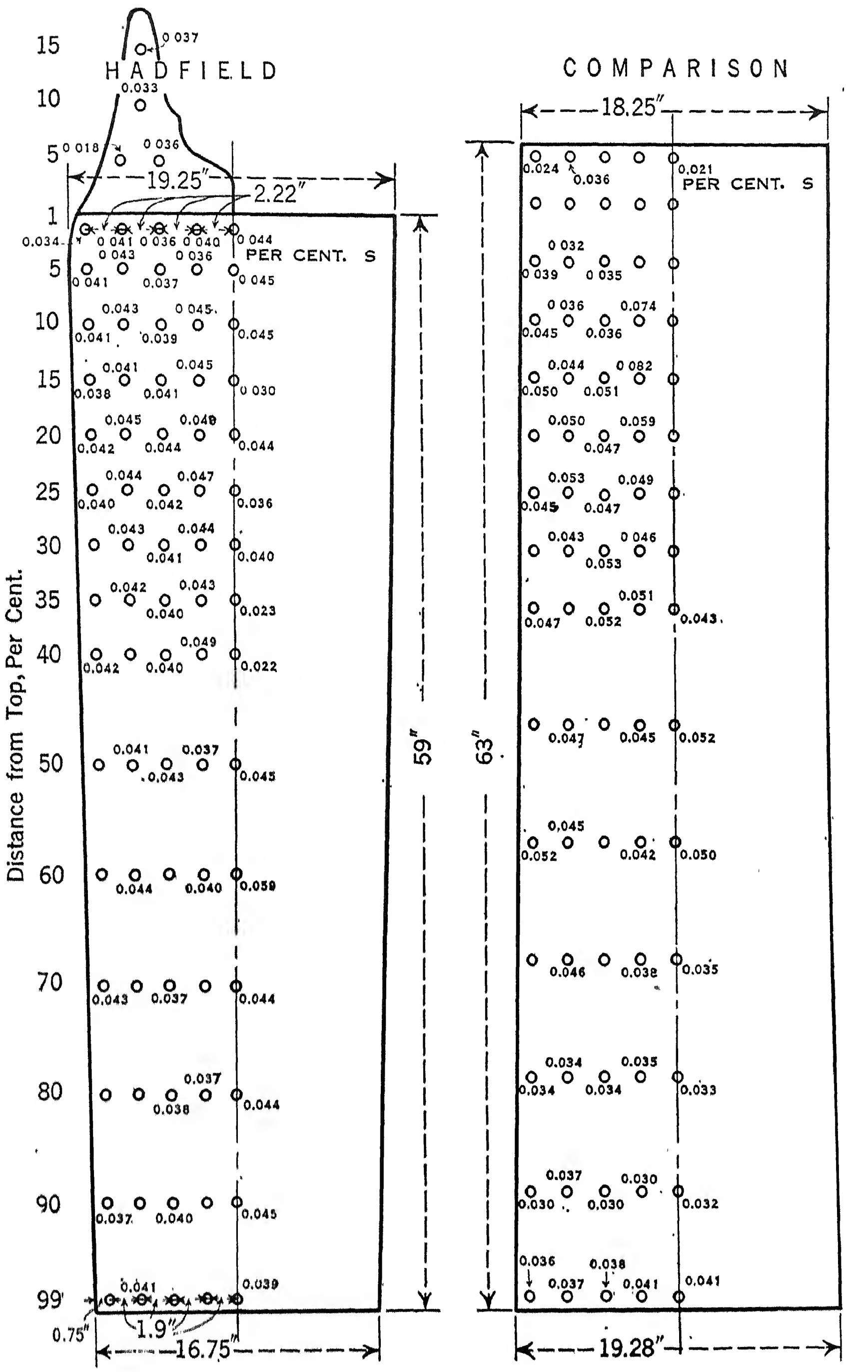
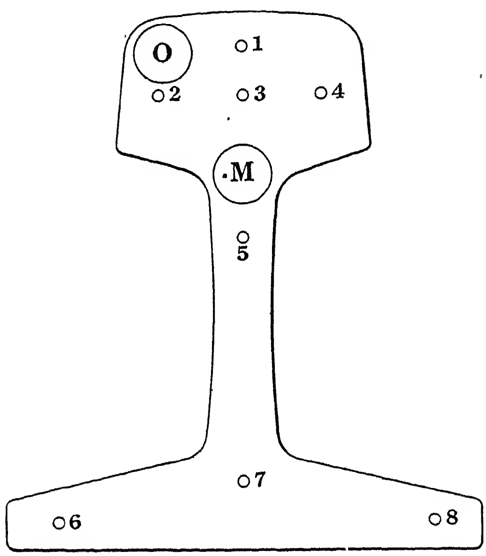


FIG. 6.—SULPHUR SEGREGATION.

tion in Table II; physical tests in Table V; and chemical analysis for carbon in Table VI in which the *O* and *M* positions refer to Fig. 7. The *O* and *M* analyses for all the elements and for the *A*, *B*, and *C* rails will be given in the complete paper.

Figs. 8, 9, 10, and 11 are characteristic sulphur prints of rail sections for each type of ingot. For all the Hadfield ingots cast large end up the rail macrostructure is very uniform even over the web for the front end of an *A* rail, as here shown. Rails from both the Hadfield ingots cast small end up show a very different macrostructure from the first group. The center streak of Fig. 11 has the appearance of nearly pure ferrite under the microscope; this corresponds also to the hardness examination of Table III. It is interesting to note that the pipe of ingot No. 6 cast in the ordinary way is not accompanied by any considerable segregation. This pipe extends well into the *C* rail even after a top bloom



O and M, positions of samples for chemical analysis.
1–8, positions of hardness tests.

FIG. 7.—RAIL SECTION.

discard of 13 per cent., confirming the characteristics of the split comparison piping ingot No. 10 of §3. In the complete paper the microstructure will also be reported in detail.

From the several tables of tests and from the metallographic examination it is seen that the percentage of ingot available for sound rails of uniform homogeneous structure is 91, 90, and 88 respectively for the specially fed ingots cast large end up, Nos. 2, 3, and 4, as compared with much less percentages of sound ingot metal usually available.

6. *General Conclusions.*—In all, nine ingots were used in this investigation; eight furnished by Sir Robert Hadfield, all but one of which had approximately the same chemical composition.

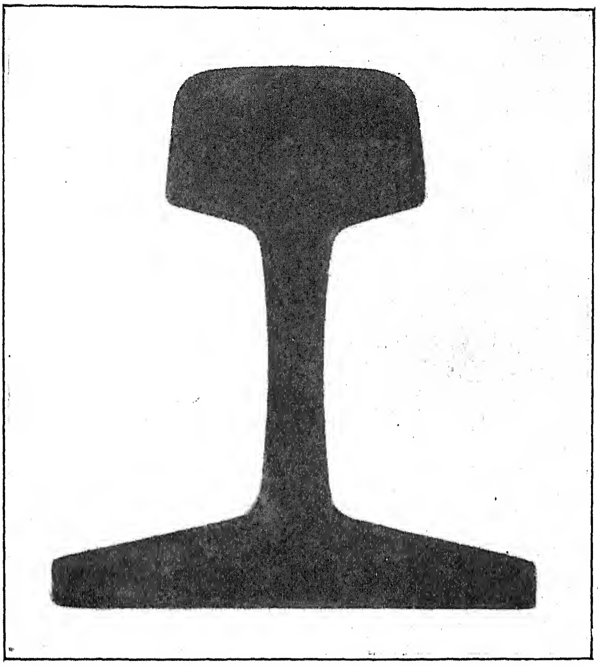


Fig. 8.—Sulphur Print of Section from Front End of *A* Rail from Ingot No. 2, Specially Fed, Cast Large End Up.

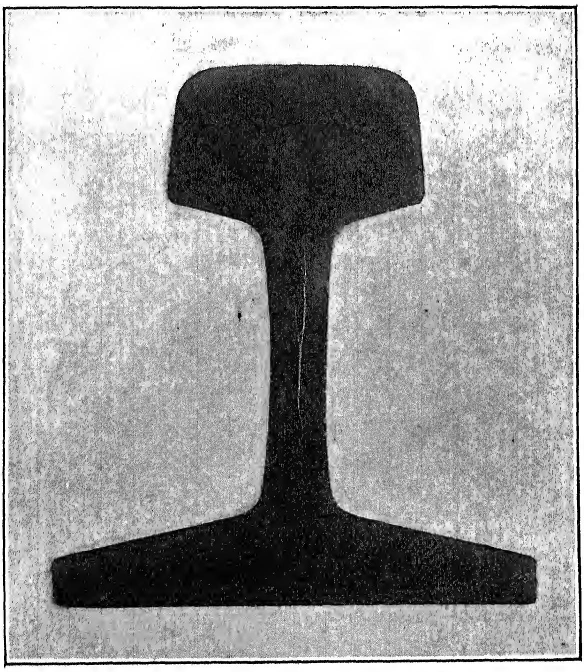


Fig. 9.—Sulphur Print of Section from Front End of *A* Rail from Ingot No. 6, Cast in Ordinary Manner from Piping Steel. The Pipe Shown Extends into the *C* Rail.

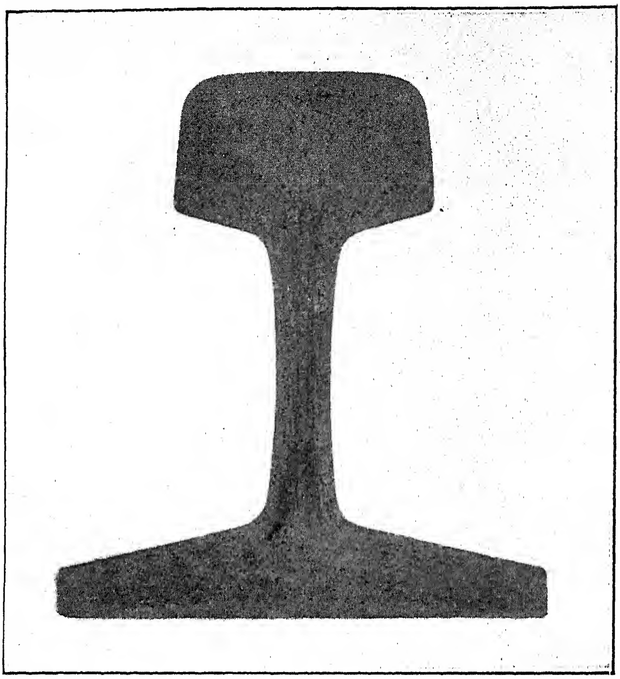


Fig. 10.—Sulphur Print of Section from Front End of *A* Rail from Ingot No. 7, Cast in Ordinary Manner from Rising Steel.

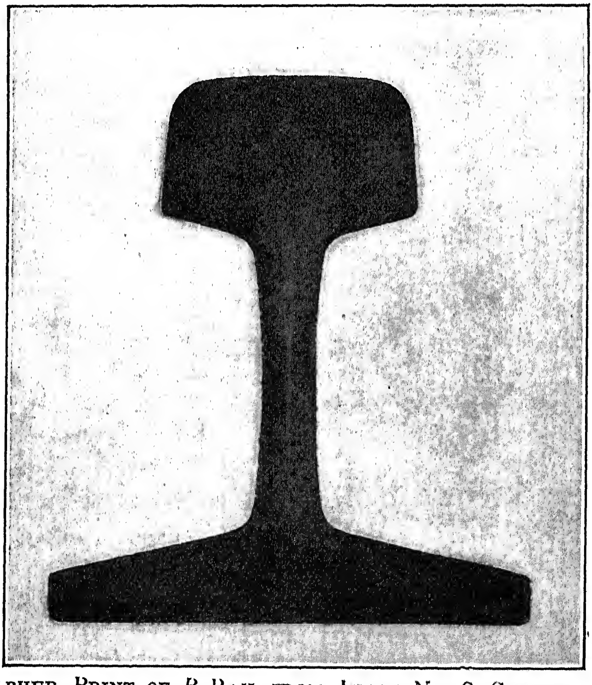


Fig. 11.—Sulphur Print of *B* Rail from Ingot No. 8, Specially Fed, Cast Small End Up.

Table III.—*Brinell Hardness Numerals*

Ingot	Rail	Hardness Numeral in Kg. Per Sq.Mm. for Positions								Mean	Average Deviation
		1	2	3	4	5	6	7	8		
2	A	250	253	250	251	272	258	258	251	255	±5.4
2	C	258	254	253	259	260	259	256	254	257	2.4
3	A	366	347	374	351	398	362	359	359	365	11.2
3	C	351	351	364	351	360	351	351	351	354	4.2
4	A	244	241	253	247	295	246	252	250	253	10.0
4	C	251	251	253	254	253	251	251	251	252	1.1
6	A	248	244	254	251	251	251	252	241	249	3.5
6	C	248	252	256	250	245	246	255	252	251	3.2
7	A	248	250	262	256	276	251	245	239	253	8.5
7	C	248	260	254	262	251	251	255	248	254	4.1
8	A	250	248	260	248	218[a]	258	260	254	250	8.5
8	C	254	258	264	251	279	261	263	263	262	5.6
9	A	248	251	251	245	236[a]	255	264	260	251	6.2
9	C	260	258	260	262	264[a]	264	264	263	262	1.9

[a] Seam through indentation.

Table IV.—*Drop-Test Results*

100-lb. P. S. section; 3-ft. supports; weight of tup, 2,000 lb.; fall, 15 ft.

Ingot No.	Blow	Per. Set	Elongation on base						
			1st in.	2nd in.	3rd in.	4th in.	5th in.	6th in.	Total
2	First..........	1.1	0.03	0.04	0.05	0.05	0.04	0.03	6.24
3	First..........	0.8	0.03	0.04	0.05	0.05	0.04	0.03	6.24
4	First..........	1.1	0.02	0.04	0.05	0.05	0.04	0.02	6.22
6	{ First, pipe at fracture.	1.0	0.02	0.03	0.04	0.04	0.03	0.02	6.18
7	First..........	1.0	0.02	0.03	0.04	0.04	0.04	0.03	6.20
	Second........	1.9	0.03	0.04	0.06	0.06	0.06	0.06	6.31
8	{ First, pipe at fracture.	1.0	0.02	0.04	0.05	0.05	0.03	0.03	6.22
9	First..........	1.0	0.02	0.02	0.05	0.05	0.04	0.03	6.21

The examination of the split ingots shows the great superiority of casting large end up and feeding by the Hadfield special process, such an ingot being physically sound and uniform throughout and practically free from chemical segregation.

TABLE V.—*Results of Tensile Tests Cut from Center of Heads of C Rails*

Ingot No.	Yield Point[a] Lb. Per Sq. In.	Ultimate Strength, Lb. Per Sq. In.	Elongation Per Cent. in 8 In.	Reduction of Area	Fracture, Per Cent.
3	98,700	177,300	9.75	27.3	100 silk, cup and cone
2	69,100	119,600	14.00	25.2	10 silk
4	69,000	118,600	12.00	26.0	10 silk
6	69,300	119,300	14.00	27.3	10 silk
7	67,000	120,600	11.75	23.6	10 silk
8	69,000	119,600	12.00	20.4	10 silk
9	68,000	119,000	11.75	20.4	10 silk

Ingot No.	Analysis[b]						
	C	Mn	P	S	Si	Cr	Ni
3	0.268	0.41	0.034	0.043	0.195	1.44	3.65
2	0.562	0.99	0.033	0.047	0.169	....	....
4	0.570	0.90	0.040	0.046	0.197	....	....
6	0.580	0.90	0.052	0.054	0.197	....	....
7	0.618	0.93	0.053	0.066	0.197	....	....
8	0.598	0.94	0.041	0.070	0.216	....	....
9	0.610	0.98	0.044	0.057	0.226	....	....

[a] Yield point found by use of dividers and running the machine back until the point was found where there was a permanent stretch.

[b] Analyses shown are from drillings taken from pulled tensile specimens.

TABLE VI.—*Carbon Analysis of Rails to Show Segregation*

O and *M* positions of Fig. 7 for tops of *A*, *B*, and *C* rails

Ingot No.	*A-O*	*A-M*	*B-O*	*B-M*	*C-O*	*C-M*	Method of Casting
2	0.540	0.612	0.560	0.556	0.570	0.588	Large end up; special feeding.
3	0.278	0.290	0.250	0.270	0.240	0.272	
4	0.550	0.640	0.560	0.580	0.612	0.580	
6	0.528	0.584	0.536	0.552	0.560	0.558	Small end up; ordinary piping steel.
7	0.550	0.648	0.570	0.598	0.578	0.568	Ordinary rising steel.
8	0..530	0.640	0.570	0.662	0.578	0.588	Special feeding.
9	0.598	0.640	0.580	0.600	0.576	0.588	

The examination of the rails shows that for ingots cast by the Hadfield process, those cast large end up will give rails of uniform quality free from flaws of all kinds; those cast small end up may show a soft region in the

web of the *A* rail but are otherwise sound. Those cast in the ordinary manner from piping or otherwise unsound steel may or may not give sound rails, depending upon the condition of the steel.

In addition to the certainty of producing all sound rails by the special-feeding process from ingots cast large end up, there is a great saving in metal and a consequent increase in percentage of sound ingot available for rails—in these experiments an average of 90 per cent. for ingots Nos. 2, 3, 4, as compared with about 50 per cent. for the piping ingot and an uncertain amount from the rising type.

The usual physical and chemical tests do not appear to give an adequate measure of the quality of rails. For example, the results of the tensile and drop tests and chemical analyses would not separate the sound from all the unsound rails here examined.

The question may be asked, Is it not worth while, in the manufacture of rails, to use only sound steel which is cast and rolled in such a manner as to make practically every rail a sound and safe one? We believe this can be done without excessive cost.

We are greatly indebted for valuable assistance from several members of the staff of the Bureau of Standards, particularly Messrs. Hillebrand, P. H. Bates, Rawdon, Witmor and Devries.

We also take this opportunity of thanking Mr. Milne and Mr. Dawson, of Sheffield, England, for their share in this research and for the care and attention bestowed by them upon the work.

Discussion

Albert Sauveur, Cambridge, Mass.—There is a definiteness and finality in the investigations that have been conducted by the Bureau of Standards and in the method of presenting the results that is most satisfactory and refreshing. A year ago Dr. Burgess and his co-workers investigated the question of critical points in carbonless iron, and they settled the question once for all. No one can now claim that the thermal point A_2 in carbonless iron does not exist.

At the last meeting of this Institute in Pittsburgh, Dr. Burgess showed the worthlessness of the shrinkage clause. To my mind, after his demonstration no one can again claim that the shrinkage clause renders any service. He also showed us that it was a practicable proposition to use pyrometers in steel work in order to ascertain the finishing temperatures of rails. No one can again come before this body and make the claim that it is not a practical proposition.

And now Dr. Burgess shows us that steel ingots can be made in a commercial way, practically free from pipe, from segregation, and from blow holes. It places again before us in a very clear manner the question of piped ingots and of segregation, a question which has been

debated for 20 years or more. I know very well the attitude of the standpatters in the matter. They claim that most of the pipe is removed in the discard, and that if a little is left it is so thoroughly welded that it does no harm. I for one am not convinced of the soundness of the argument. I do not believe there is any one present who would claim that, given an ingot free from pipe, free from segregation, and free from blow holes, better rails will not be made out of that ingot than out of the ordinary piped or segregated metal which is used by almost all of our steel mills. In my opinion, if we give our sanction to the casting of steel in the shape of piped or segregated steel ingots, we are guilty. If we give our sanction to the rolling of rails from such metal, we are guilty. It is our duty as experts, as steel makers, and as steel consumers to make and to use the best steel obtainable on a commercial basis, for the manufacture of rails.

Now, we have been shown that sound ingots can be produced from which superior rails can be made. If we do not insist upon these sound ingots, then we fail in our duty. I think that the art of casting steel ingots has advanced to such a point that the casting of steel ingots piped and segregated ought no longer to be tolerated.

I do not mean to say that the Hadfield process is the only one that will do it. Indeed, I know that there are other processes, and that at some mills methods are used which will practically prevent the casting of piped and segregated metal, but I claim that the practice should become general.

Henry M. Howe, Bedford Hills, N. Y.—I think the ethical question now comes up—is it not incumbent upon us in every reasonable way to bring every reasonable pressure to prevent the continuation of peril to the lives and safety of passengers on railways from the use of piped and segregated rails?

* Can the authors tell us in how great a proportion of cases the suppression of segregation and piping is as thorough as it is in the cases which they show? Or can they tell us how large their experimental basis of observation is, to the end that we may form an opinion as to how large the proportion of cases is in which these very great benefits may confidently be expected?

I of course have in mind the experience which so often has come to us, that an apparently very great improvement occurs in certain cases, but is not reproduced regularly in all, or is reproduced in only a certain proportion of cases.

Henry D. Hibbard, Plainfield, N. J.—Doctor Burgess has stated that ingots cast by the Hadfield method are always right, but it should be understood that no method of casting will make good ingots unless the

* Communication to the Secretary. Received Mar. 25, 1915.

steel is right before it goes into the molds. Beside having the proper composition as to desired ingredients it must have full tendency to pipe, and all impurities within permissible limits. These impurities are the soluble phosphide and sulphide of iron, the insoluble oxides, silicates and sulphides of iron and manganese, and the gases, carbonic oxide and hydrogen. When rail steel meeting these requirements is cast into pipeless ingots, good, sound, serviceable, trustworthy rails can be made therefrom.

Joseph W. Richards, So. Bethlehem, Pa.—I think that both this paper and the preceding one should have a short paragraph inserted in the paper saying how the temperatures were determined, the kind of apparatus and the way it was used, because probably these results will be duplicated by others, and there may be misunderstandings, because of using different methods of measuring the temperature.

F. W. Wood, Sparrows Point, Md.—(communication to the Secretary*).—While agreeing fully that sound ingots with a minimum of segregation can be produced by Sir Robert Hadfield's method of casting, if properly conducted, it is an open question to what extent the total of rail failures in service would be reduced by its general adoption.

The statement that ingots so cast will give rails free from flaws of all kinds, is assumed to refer to interior flaws.

Obviously, exterior seams, the result of breaks in the skin of the ingot from shrinkage while in the mold or from the early passes in the blooming rolls, will not be affected. Likewise, inclusions of slag or other foreign matter, near the outside of the ingot, presumably the starting points of split heads, may be expected to be as numerous as in ingots cast in the usual way.

The obscure and very dangerous type of failure, the transverse fissure, which develops in rails from all parts of the ingot, has not been connected with piping or segregation, the two irregularities the Hadfield process is claimed to remedy.

Records from railroads with heavy traffic show breakages of flanges and through the entire section of the rail to be as numerous in rails from the lower two thirds of the ingot as from the upper, and consequently not mainly due to pipes or irregularity of composition.

It is not the purpose of these comments to question the desirability of sound and non-segregated ingots as a factor in securing the best possible service, but rather as supporting the writer's opinion that any positive relief from rail failures, especially of dangerous types, must be looked for in other directions.

George K. Burgess.—I will say, in regard to Professor Richards's statement, as a matter of record, that the temperature measurements in

* Received Feb. 13, 1915.

this communication were made in exactly the same way as the temperature measurements referred to in the previous communication, on finishing temperature of rails, which I presented at the last meeting.

Of course, the statements made by Mr. Hibbard, as to first having sound steel, are unquestionably acceptable from all points of view.

The several factors contributing to the production of unsound finished products, to which Mr. Wood calls attention, are undoubtedly vital, but the existence of these various imperfections should serve as an incentive to eliminate each cause of unsoundness as opportunity offers.

Henry M. Howe (communication to the Secretary*).—On p. 866 it is said that "the comparison ingot, if rolled into rails, should have about 50 per cent. discard." This seems to require some explanation. A discard of a little over 20 per cent. would have removed all the metal containing more than 0.104 phosphorus, and the remaining nearly 80 per cent. would have had no metal with more than 0.096 per cent. except for a rather short distance along its axis, where it rises to 0.104 per cent. With an average phosphorus content of 0.09, an enrichment in the very axis up to 0.104 certainly is not serious.

An examination of the segregation indicates that the advantage of the Hadfield casting is not in a reduction of the degree of segregation but in confining segregation to the very top of the ingot, as is natural. For instance, the greatest Hadfield enrichment in carbon in Fig. 4 is from 0.59 to 0.79, a gain of 0.20, or 34 per cent. The enrichment in carbon of the comparison ingot is from 0.46 to 0.59, a gain of 0.13, or 28 per cent.

But there is this difference, that the Hadfield 0.79 is confined to about 5 per cent. of the upper part of the ingot, whereas the 0.59 of the comparison ingot is somewhere about 15 per cent. from the top.

The hardness numbers in Table III are not so favorable to the Hadfield metal as might have been hoped. The *A* rail of the No. 3 ingot has a greater average deviation than any of the rails with which it is compared, and the next greatest deviation comes in the Hadfield *A*4 rail, 10 per cent.

A result of interest is the comparison of the piping steel, No. 6, and the rising steel, No. 7, in Tables III and VI. The piping steel has a smaller deviation in hardness than the rising, and a much smaller deviation in carbon content, namely, 0.528 to 0.584, or 0.056, whereas the rising steel has a deviation from 0.550 to 0.648, or 0.098, nearly twice as much.

* The belief that the grain size reached increases with the total time available in heating, at heat, and in cooling, is supported by the data of Burgess and his collaborators,[2] although they put a different inter-

*Received Feb. 24, 1915. † Received Mar. 27, 1915.

[2] *Technologic Paper No.* 38, *U. S. Bureau of Standards*, Observations on Finishing Temperatures and Properties of Rails, p. 41 (1914).

pretation on them. They find that the average number of grains per square inch of 90-lb. and 100-lb. rails is 24,000, whereas that of 72-lb. and 75-lb. rails finished at the same temperature is 42,800. Thus the thinner and more rapidly cooling rails have developed grains only about half as large as those of the larger and hence more slowly cooling rails.

Further, they find the average number of grains per square inch for all their rails is only 26,200 for the thick and slowly cooling head, against 44,800 for the web and 36,900 for the base. These differences are no doubt referable in part to the higher finishing temperature of the head, but they certainly support the belief that the necessarily slower cooling of the head also has favored grain growth.

The comparison of the web and base is especially interesting. As they leave the rolls the base certainly ought not to be materially hotter than the web, and hence, if the grain size depended solely on the finishing temperature, its grain size ought not to be greater than that of the web. But on the hot bed the cooling of the web is likely to be materially faster than that of the base, for the reason that the base of each rail rests against the thick hot head of the next, whereas the web is exposed freely to the air on both sides. Hence the smaller grain size of the web than of the base seems clearly due to the more rapid cooling of the web, the shorter time available for grain growth.

Sir Robert Hadfield (communication to the Secretary*).—In answer to Professor Howe's remarks, I beg to point out that on 70,000 tons of ingots, weighing from ½ ton to 2½ tons each, made by the sound-steel system described in our paper, and representing tens of thousands of separate ingots, *not a single defective ingot has been found.*

On this question of soundness there have never been truer words uttered than those of Professor Sauveur. It is hardly necessary for me to add anything to his remarks except to say that it is possible to guarantee that not merely on small numbers but on any quantity of ingots perfectly sound steel can be obtained; in other words, the reproach of unsound, defective, and cracked material has been done away with.

As regards Mr. Wood's rematks, I believe he will find that sound ingots such as those now described will give far superior steel. He seemed to have overlooked that the system embodies not only the manufacture of ingots by means of which waste is avoided, but that equally, if not more important, is the fact that the steel is "sound," free from segregation and piping. Therefore quite apart from the great advantage of less discard, the sounder steel obtained should give relief from unaccountable failures met with in rails and other steel products. If, however, the manufacturer still insists upon continuing to make, and the user goes on using, inferior and unsound steel, then there can be only one result: namely, that there

* Received May 17, 1915.

will be a continuance of breakages and failures of different kinds, in the various articles into which steel ingots are made.

No steel is perfect and no method will avoid every trouble, but without doubt the failures such as those hitherto experienced can practically be avoided.

With regard to Professor Howe's remarks on the question of the enrichment in carbon, he seems to overlook that some of the steel in the upper portion of the ingot in question was probably carbonized with the charcoal used to keep the upper portion hot; consequently at this point the percentage of this element must necessarily be higher.

Are the Deformation Lines in Manganese Steel Twins or Slip Bands?

BY HENRY M. HOWE AND ARTHUR G. LEVY, NEW YORK, N. Y.

(New York Meeting, February, 1915)

§1. INTRODUCTION.—Any given piece of metal is made up of a very great number of *grains*, usually microscopic, each of which is a perfect crystal save only in outward form, with *cleavage planes* of low cohesion, geometrically arranged quite as in the familiar non-metallic crystals.

The plastic deformation of such a mass occurs chiefly through the *slipping* of the crystalline blocks of which each grain is composed along these cleavage planes, causing steps called *slip bands* to form on a previously polished surface. In this slip, the metal immediately adjoining each of the cleavage planes along which the slip takes place is thought to pass extremely rapidly through a very *mobile* state into the *amorphous* state, in which it is harder than the remaining still crystalline metal between the amorphous layers thus formed along the slip planes. Whether the slip planes below the surface in ferrite can be detected by cutting, polishing, and etching sections is in dispute; but if this detection is possible at all, it is usually made impossible by heating the metal even gently. In short, the change along the slip planes does not persist through heating, and it is thought not to persist through time even at the room temperature, nearly all the amorphous metal recrystallizing.

The network in Fig. 22 and the parallel N.–70°–E. lines in Fig. 21 are slip bands in copper.

When plastically deformed metal is heated, certain parts of certain grains may *twin*, that is, their component crystalline units may rotate into a position, or more exactly into an orientation, symmetrical with the initial orientation and hence with that of the remainder of that grain. As seen in a microsection of a metal the twinned areas usually have parallel sides. The existence of twinning is recognized in metallic sections either by the presence of such parallel-sided areas differing in brightness from the adjoining metal, especially when seen under oblique light after etching, or more surely by the zigzagging of the slip bands which form when twinned metal is itself deformed again. Hence the usual procedure in developing twins is to deform plastically so as to cause the twinning tendency; to heat so as to allow the tendency to assert

itself; to polish so that slip bands may next be made visible, and to deform again so as to create slip bands. On this polished surface such slip bands change direction sharply on entering each twinned area, and revert to their initial direction on leaving it.

Twins made manifest in this the usual way may be called *annealing* twins, to distinguish them on one hand from congenital twins formed in solidification or transformation, which do not concern us in this paper, and on the other hand from *mechanical* twins, such as the Neumann lines are supposed to be. These are narrow bands which form in alpha iron or ferrite, usually only on deformation by shock, though we have succeeded in developing them in silicon steel by quiescent deformation in a vise. They are caused by the deformation without the need of subsequent annealing.

The parallel-sided band, 4, in Fig. 2, and the areas 1 to 7 in Fig. 1, are twins. Note how the manganese steel lines in Fig. 1 have two alternative directions, following one of them in the odd-numbered twinned areas, and the other in the even-numbered areas. And note how the slip bands in Fig. 21 change from N.–70°–E. to nearly horizontal and back again as they pass through the twinned area *abcd*.

Because twinning usually occurs in other metals, not during the plastic deformation but during a heating which follows it, and for other reasons, it cannot be regarded as the usual mechanism by means of which plastic deformation occurs, but rather as a result of deformation.

On a previously polished surface of Hadfield's austenitic manganese steel quenched from 1,000° to 1,100° plastic deformation develops lines which though they look much like slip bands, as in Figs. 1, 2, 6, 7, 8, and 10, yet differ from most slip bands not only in reappearing persistently on repolishing and re-etching as in Figs. 3, 4, 11, 12, and 15 but in persisting through considerable reheating, and indeed then becoming very much more prominent, through the precipitation of cementite along them as in Figs. 5, 9, 13, and 14.

What are these lines? According to Osmond and Cartaud[1] they are extremely numerous twins. The present paper, after outlining in Section 3 and 4 the life history of these lines, offers in Sections 13 to 17 evidence and reasoning tending to show that they are slip bands comparable with those which form in other metals and alloys. Meanwhile let us call them simply the *manganese steel lines*, so as to avoid committing ourselves to any theory by means of a more indicative name.

§2. Twins of the usual kind have not, so far as we know, been shown hitherto in manganese steel, nor have we succeeded in developing them

[1] *Journal of the Iron and Steel Institute*, vol. lxxi, (1906, No. III), p. 468. Such multiple or "polysynthetic" twinning does indeed occur in non-metallic native minerals. Compare Iddings, *Rosenbusch's Rock-Making Minerals*, Plate 25, Fig. 6 (New York, 1889).

except by very severe deformation, followed as usual by annealing, polishing, and re-deforming. But by this means we have developed a great number of very typical twins. Two twinned fields in this alloy are shown in Figs. 1 and 2. These represent metal from the immediate neighborhood of the fracture of a tensile test piece, which after rupture was heated to 1,000° for 1 hr. to develop the twins, quenched, polished, and then again deformed so as to develop the zigzagging lines. In Fig. 1 the lines follow a common direction throughout the four odd-numbered areas, and another common direction throughout the three even-numbered ones. In Fig. 2 they follow a common direction throughout the three odd-numbered areas, but not in the two even-numbered ones. The sides of these areas are in general parallel, especially those of area 4 of Fig. 2. In Fig. 1 the brightness of the nose of the tread or boundary between areas 4 and 5 and the darkness of the boundary between 5 and 6 are typical.

In strong contrast with the difficulty of causing typical twins in Hadfield's manganese steel is the great ease with which they form in an iron alloy containing 33 per cent. of manganese, and represented to be free from carbon. This was made for one of us by the Goldschmidt Thermit Co., of New York. In the etched section of a cast bar of this alloy, prepared without any deformation of which we are aware save that of sawing it open, we found large perfectly typical twins after reheating to 1,100° and quenching.

§ 3. THE MANGANESE STEEL LINES.—*The progress of their development*, as traced on the previously polished surface of a wedge gently compressed,[2] is shown in the series of Figures from No. 16 to No. 19, Plate 4.

At the beginning of the deformation, shown in Fig. 19, there are a few parallel lines, a gentle undulating disturbance of the whole surface, and

[2] This method we have borrowed and modified from Frémont, Mesure de la limite élastique des métaux, *Bulletin de la Société d'Encouragement pour l'Industrie Nationale*, vol. cv (1903, II), p. 363. It was followed by Osmond, Frémont, and Cartaud in their classical investigation, Les modes de déformation, *Revue de Métallurgie*, vol. i, p. 27 (1904). These investigators used a truncated pyramid, but for simplicity we have replaced this with the truncated wedge shown in Fig. 24, ½ in. high from base to truncated top, ⅜ in. thick, ½ in. by ⅜ in. at the base, and ¼ in. by ⅜ in. at the top. After polishing one of the trapezoidal faces of the wedge so that the deformation lines might be seen there, we compressed it between polished and greased steel plates, with pressure applied at the wide end or base and at the truncated top or narrow end. Strangely enough there was in most cases, in manganese steel, copper, and other metals, a band almost free from deformation, running along the upper part of the wedge, probably because of friction with the compressing plate, in spite of the greased contact between them intended specially to prevent this result. Below the region where the dulling of the polished surface ceased, and near the base, came a new region of slight deformation, the arc of a circle of long radius.

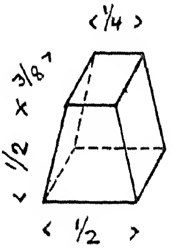


FIG. 24.

the outlining of some of the grain boundaries, all three phenomena reproducing closely what occurs in other metals except alpha iron. Then these parallel lines become more marked, and are crossed by a conjugate set, as in Fig. 18. Next they become both more numerous and more prominent, as in Fig. 17. Under still greater deformation the surface distortion or ruffling becomes so great that it tends to mask these lines, as in Fig. 16.

The parallel lines in manganese steel, and the slip bands in copper, become visible at about the region where the undulation of the surface can first be recognized, and hence under about the same degree of deformation. Judged by means of the accompanying degree of surface undulation, the manganese steel lines appear if anything a little earlier than the slip bands in copper. Even where the surface deformation is very marked there are undulated areas in which no lines can be traced under a magnification of 50 diameters, and but a few under a magnification of 500 diameters. Nevertheless the whole of the undulation may be accompanied and accounted for by slip bands so minute as to escape detection.

§4. THEIR CHARACTERISTICS.—Whether caused by shock or by quiescent deformation they are in parallel sets, sometimes in one direction only in each grain, as in Figs. 3, 7, 11, and 19, but more often in two and sometimes even in three directions, as in Figs. 4 and 5. These directions were identified by Osmond and Cartaud as representing the octahedral cleavages.[3] They often recall martensite, when, as in Figs. 4 and 5, two sets of them are of about equal prominence. In their incipiency they may run but part way across a grain, as in Figs. 6 and 19, but under greater deformation they usually extend completely across the grain in which they occur, as in Figs. 3, 5, and 10. Though on reaching a grain boundary they generally stop, as in Figs. 3, 5, and 10, they may continue across grain boundaries, sometimes without changing direction, as in Fig. 13, sometimes deflecting sharply, as at the boundary *ab*, in Fig. 11, and at the boundaries between grains *c* and *e*, and *f* and *g* in the same figure. They may cross the dendrites without the least deflection, as in Fig. 8. They are usually very nearly straight, or curved smoothly and gently, but occasionally they wave like lamellar pearlite, probably in cases of great deformation, as just above *e* in Fig. 11.

They very rarely fault each other sharply as slip bands in other metals and Neumann lines in alpha iron often do, perhaps because, for given deformation, they are so numerous that the fault at any one intersection is so slight as to escape detection. Indeed some at least of what seem at first to be faults on closer examination prove to be twinned areas.

A very slight deformation suffices to cause them. Thus, though they are lacking at least usually in castings as taken from the sand, they may

[3] *Loc. cit.*

be made extremely prominent by the slight unintentional deformation which occurs when a casting cut almost completely through, is then broken in two.

As they form the metal becomes somewhat ferro-magnetic, and increases greatly in hardness. Thus the Brinell hardness near the tensile fracture shown in Fig. 12 was 340, whereas in the undeformed part of the same test piece it was only 217.[4] This magnetization indicates that some of the metal is transferred to the alpha state. The hardness is to be referred at least in part to the amorphousness of this alpha iron caused by the slip and perhaps in part to the cementite which may be precipitated simultaneously as explained beyond.

Thus Fig. 12, showing the longitudinal section through the fracture of a tensile test piece, suggests that its white component is alpha and in part amorphous iron formed along the slip planes on all sides of the dark rhomboids of the residual crystalline and presumably gamma iron. A larger magnification, as in Fig. 15, tends to confirm this interpretation.

§5. The Chief Reasons for the Usefulness of this Alloy are probably its ability to retain not only continuity but strength through an extraordinarily great degree of amorphization, and the great ease with which this amorphization occurs, two aspects of the same thing. Its surface flows very easily under strong pressure, as for instance that which it meets in the jaws of stone crushers, but in this very flow it hardens greatly and rapidly, so that a relatively hard wearing surface forms, integrally united with a ductile back. Moreover, this surface is self-renewing, for as fast as a given layer is worn away the next, as it thus becomes the surface, simultaneously hardens under the deformation which it at once undergoes.

§6. The Precipitation of Cementite along thsee lines is evidently one cause of their apparent persistence.

The transfer of the iron from the gamma to the alpha state caused by the deformation may perhaps itself precipitate cementite, which is nearly or quite insoluble in alpha iron though abundantly soluble in gamma iron, but we cannot be certain yet that the cementite is able at so low a temperature to escape from solution. However this may be, the heating certainly should enable this cementite to precipitate, and as the temperature rises and the mobility increases, to coalesce into visible masses. This same mobility should allow both the pro-eutectoid cementite, the precipitation of which has been suppressed in the quenching, to precipitate, and the gamma iron itself to change into alpha, with consequent precipitation of the remaining cementite.

[4] Hadfield and Hopkinson found the Brinell hardness of a tensile test piece after rupture 540, against about 220 in like material before straining. (*Journal of the Iron and Steel Institute*, vol. lxxxix (1914, No. I), pp. 112 and 124, and *Trans.*, l, p. 486 (1914).

Each block of crystalline metal, which we may picture to be represented by the dark rectangles of Fig. 12, being surrounded on all sides by amorphous metal, would now act as an independent crystal, and expel to its outside all the cementite precipitated within it. This cementite should thus become assembled along the very cleavages in which the slip has created amorphous iron, and because these cleavages are octahedral, it would form the familiar Widmanstättian figuring, so prominent in Figs. 5, 13, and 14. The thickening of the manganese steel lines on reheating strained specimens is evidently due to this assembling of cementite along them, as is shown most strikingly in Fig. 14.

This crowded migration of cementite would be likely to sweep along with it any inert matter, or sonims, such as the slag and manganese sulphide so often present in this steel.

§7. Cementite, not Martensite.—This cementite was very naturally taken by Sauveur[5] for martensite, which it often simulates closely, both of them lying Widmanstättianwise along the octahedral cleavages like the amorphous iron. It is shown to be cementite by its blackening with sodium picrate;[6] by its assembling, under favorable conditions, into a network closely like that of the pro-eutectoid cementite of hypo-eutectoid carbon steel; and by its being apparently identical with the corresponding cementite network which forms in this manganese steel, as shown in Figs. 5 and 13.

§8. Additional Cause of the Persistence of These Lines.—If the reappearance of these lines after reheating were due solely to the assembling of cementite along them, then they ought to thin out progressively on heating to temperatures above A1 and then quenching, and to be effaced progressively on heating to above SE and quenching, so that the proportion of these lines which reappear ought to be the smaller the higher and the longer the heating is. As regards temperatures between A1 and SE our scanty observations do not disagree with this expectation. But the reappearance of these lines after heating to 1,000° and quenching, which certainly was unforeseen, indicates that their persistence is connected with something in addition to the presence of cementite, which even in this sluggish material ought to re-dissolve completely in a stay of 1 hr. at 1,000°, and to remain dissolved during the quenching if, as is probable, the line SE is below 1,000° for this material.

The persistency of this specific Widmanstättian figuring tallies with its persistency under other conditions,[7] and with that of the initial

[5] *Trans.*, l, p. 501 (1914).

[6] Howe, *Trans. Faraday Soc.*, vol. x (1914); *Engineering*, vol. xcix (1915), p. 87.

[7] We showed that this structure has extraordinary stability in a 0.40 carbon steel, suffering no important breakdown on staying either for 137 hr. at 650°, or for 21 ¼ hr. at between 725° and 740°, though in this latter case the very strong cementite network structure of a 1.14 carbon steel beside the Widmanstättian steel broke down com-

dendritic casting or ingot structure,[8] both of which persist through treatments which at first would be expected to efface them. In some cases this persistency represents simply a defiance of surface tension, in others it may be referred to the co-concentration of minute quantities of impurities such as phosphorus, which diffuses very slowly, or of sonims which do not diffuse. In this present case, the persistence of the manganese steel lines through a long sojourn at 1,000° and quenching may be due to the latter cause, lines of impurities once swept along by the migrating cementite into the cleavage planes where the amorphous iron formed, remaining undiffused long after the cementite which dragged them there has diffused away and been reabsorbed, and the amorphous state has reverted to the crystalline state, and the gamma iron to alpha.

§9. The Persistency of the Manganese Steel Lines.—Some of these manganese steel lines reappear after repolishing and re-etching, showing that the metal, along the planes of slip throughout the mass, has undergone some change which causes it to react with the etching reagent differently from the undisturbed remainder of the metal. Rosenhain[9] indeed finds that in other metals, if the slip is great, it can be detected in this way, by repolishing and re-etching; but that the change thus detectable by etching effaces itself very rapidly on slight heating, and slowly even at the room temperature, whence he infers that the metal reverts spontaneously from the amorphous state caused by the deformation to the crystalline state, and thus becomes undistinguishable from the remainder of the mass. Strauss,[10] too, finds that steel which has been nitrogenized by heating in ammonia, when deformed even slightly and unintentionally and then polished and etched, shows martensitic markings, which therefore should represent a detectable change induced along the slipping planes by the deformation.

But the manganese steel lines persist even through reheating. Or more accurately, after the deformed metal has been reheated, to repolish and re-etch it develops lines, shown in Figs. 5, 13, and 14, following the directions of those found immediately after deformation, as for instance those of Figs. 4, 10, 11, 12, and 16 to 19 inclusive. Indeed the lines found after reheating may be much more prominent and thicker than those found before. Note how conspicuous the white masses in Fig. 14 are after reheating, repolishing, and re-etching. On a smaller magnification, as in Figs. 5 and 11, they are seen to correspond to those

pletely. Notes on Divorcing Annealing and Other Features of Structural Coalescence in Iron and Steel, *Proceedings of the Cleveland Institution of Engineers*, July, 1914, pp. 234 to 243, and remarks by Stead on p. 252.

[8] Engineer N. T. Belaïew, *Révue de Metallurgie, Mémoires*, vol. ix, No. 9, p. 647 (Sept., 1912).

[9] Metals, Crystalline and Amorphous, *Engineering*, vol. xcvi, No. 2493, p. 510 (Oct. 10, 1913).

[10] *Stahl und Eisen*, vol. xxxiv, No. 50, p. 1814 (Dec. 10, 1914).

found after etching the deformed metal without heating it, as in Fig. 4; and these in turn are seen to correspond to those which form on the polished surface during the deformation itself, as in Fig. 7.

The temperatures after heating to which these lines reappeared on repolishing and re-etching in our experiments are 400°, 550°, 575°, 600°, and 800°. Some of these lines could be traced even after a stay of 1 hr. at 1,000° followed by quenching, polishing, and etching. It was not the quenching that caused them in this case, because we find that polishing and etching does not develop them in a specimen quenched from 1,000° without previous deformation.

That they persist at the room temperature is to be referred to the formation of alpha iron along them, which of course has no tendency to revert to gamma iron. But, on heating, this gamma iron itself ought to change over into alpha, and the amorphous iron ought to revert to the crystalline form, so that here another cause must be sought for the persistency of these lines.

§10. Cementite and Rupture.—This pro-eutectoid cementite network forms the path of rupture in manganese steel, as is shown in Fig. 9, even more prominently than in hyper-eutectoid carbon steel, and is probably extremely embrittling in both alloys.

§11. Groupings Which Simulate Twinning.—There are two distinct sets of groupings of these manganese steel lines which at first look like twinning, but on further examination may prove not to be. The first set represents the intersection of these lines at grain boundaries, the second represents an alternation of the relative prominence of two conjugate sets of these lines.

§11A. Intersections at Grain Boundaries.—In Fig. 11 the intersection *ab* of two sets of lines certainly looks like a boundary between two twinned areas, and not between two adjoining grains, for not only is the proportion of the lines which continue past *ab* from one area into the other much greater than the proportion of slip bands which usually cross a grain boundary, but also the change of direction is much greater than is usual in slip bands which cross grain boundaries. Yet the shape of the two areas on either side of *ab*, while very far from that of the parallel-sided twin areas, is exactly that of the usual grain areas. This is even more strikingly true of such areas as *c* and *d*, and of *f* and *g*, yet these manganese steel lines cross the boundaries between each of these pairs of areas with a marked deflection. Because so many of these areas are thus exactly grain shaped, and because this alloy has occasional twinned areas with the characteristic parallel-sided twin shape, and wholly unlike these grain-shaped areas, we should take these latter for grains, and infer that in this alloy an unusually large proportion of the slip bands continue across the grain junctions even when, as at *ab*, this implies an apparent great change of direction, remembering that the apparent change depends

wholly on the inclination of the plane of the section to the planes of slip.[11]

§12. TWINNING SIMULATED BY A CHANGE IN THE RELATIVE PROMINENCE OF TWO CONJUGATE SETS OF MANGANESE STEEL LINES.—Both in the polished but unetched and in the polished and etched sections of plastically deformed manganese steel, there are very often two conjugate sets of lines, certain individuals of which stop or lose prominence on intersecting a line of the conjugate set, or lose prominence on entering a certain area. Thus in Fig. 6 two lines running from *c* to *d* at first look as if they stopped at *d* and changed direction, whereas in fact they continue their direction down beyond *d*, but become fainter at the left of *d*. So the lines running S.–60°–W. from *f* at first seem to stop at *e* and change direction, whereas in fact they extend beyond *e* but lose

[11] Humfrey indeed asserts that slip bands in ferrite do not cross grain boundaries unless they can do this without change of direction (*Carnegie Scholarship Memoirs, Iron and Steel Institute*, vol. v, 1913, pp. 91, 92). But his Fig. 7, Plate XI, shows at least four slip bands which deflect more than 90° in crossing a grain boundary. Moreover, the apparent angle of deflection must need vary greatly with the direction of the plane of the section, as is shown by a very simple experiment, which we commend to our readers as helping them to grasp these general conditions. Cut out of a sheet of stiff paper an isosceles-triangle arrow head, the two sides of which meet at an angle of about 70° like the lines on either side of the intersection *ab* of Fig. 11. Set an open book on end on your desk, insert the point of the arrow head, with its base horizontal, between the leaves, and open or shut the book till the arrow head just fits the opening between its leaves. If the bisectrix of the triangle is held horizontal, the leaves will of course fit the sides of the arrow head when they meet at this same angle, 70°. If the book is now opened gradually wider and wider, we can continue to make the sides of the arrow fit the space between the leaves by simply raising its point and lowering its base, theoretically till the book is open wide so that its two covers are in the same plane.

Now let the arrow head represent the plane of our microsection, and the leaves of book represent the direction of two slip planes which approach each other across adjoining grains, and intersect at the grain boundary, the back of the book Let the book be nearly wide open, the angle between its covers being 170°. The sides of an arrow which if held horizontally is to fit between these leaves would have to meet at this same obtuse angle, 170°. If, now, the arrow is inclined more and more by raising its point while keeping its base horizontal, it will have to be trimmed off first so as to be less obtuse, then so as to be a right angle, and then to a more and more acute angle, in order to remain always just fitting in between the leaves, which remain at an angle of 170°. If we replace the arrow by the plane of our microsection then if we incline it more and more from the horizontal, the angle at which the leaves will cut it will in like manner become first less obtuse, then a right angle, and then a more and more acute angle. In short, the apparent angle between the slip planes on the two sides of a grain boundary varies greatly with the inclination of the plane of the microsection to those planes. Thus in two adjoining grains A and B, two slip planes, which in fact are within 10° of being parallel so that slip along one of them in grain A might easily cause slip along the other in grain B, if cut by a microsection which is strongly inclined to both may seem to make a very sharp angle, so that the continuation of slip bands across the grain boundary looks as if the direction of slip changed by say 100° as at *ab* in Fig. 11 though in fact it changes by only 10°.

prominence. Again, at the imaginary line *ab* a great number of the lines running S.–60°–W. seem to twin into corresponding lines running N.–45°–W., as if *ab* were the boundary of a twinned area. Yet if we extend *ab* a little higher up we see that these two sets of lines in fact continue beyond their intersection. So in the etched Fig. 4 certain areas running N.–80°–E. seem to be twinned areas, for in the first, third, fifth, etc., the prominent lines run N.–40°–W. whereas in the second, fourth, sixth, etc., they run N.–40°–E. But on closer examination these two sets of lines, N.–40°–W. and N.–40°–E., are seen to co-exist in a larger part of each of these areas, with the difference that in the odd-numbered areas the N.–40°–W. lines are the more prominent, whereas in the even-numbered areas the N.–40°–E. lines are. In short, though these cases certainly suggest strongly the true staircase twinning effect of Fig. 1, in which lines running in a given direction change abruptly to another direction on entering a parallel-sided area, yet the co-existence of both sets of these lines in a large part of these areas argues that these are cases not of twinning substitution of one direction for another, but only of a change in the relative prominence of two sets of lines.

§13. Are the Manganese Steel Lines Twins or Slip Bands?—We find five reasons for interpreting them as slip bands.

(1) With progressively increasing deformation, the first appearance and the development of these lines are like those of the slip bands in other metals, as we find by applying to iron, copper, and lead the truncated wedge experiment outlined in Section 3. This resemblance is shown by a comparison of Figs. 16 to 19, which illustrate the development of these lines in manganese steel, with Figs. 20 to 23, which illustrate the development of the slip bands in copper.

Save that copper roughens more than manganese steel, one metal might often be taken for the other, so closely do the manganese steel lines and the copper slip bands correspond in their development. Compare for instance Figs. 18 and 22. The manganese steel lines curve sharply at their intersections like those of copper, and they and the copper lines deviate from straightness in about the same degree, as is seen by holding a straightedge along lines *ab* of Fig. 18 and *a′b′* of Fig. 22.

These lines in copper are certainly slip bands. Those in manganese steel start and develop as closely like those in copper as could be in two different metals. The differences are rather such as we expect between identical phenomena in different substances, such as the difference between the waves in oil and those in water and sulphuric ether, than such as we find between radically different phenomena.

§14. (2) These lines in manganese steel bear to the twinned areas the same relation as regards their order of magnitude that slip bands bear to twinned areas not only in copper, silver, and lead, but also in 25 per cent. nickel steel, the substance most closely comparable with manganese

steel, differing from it primarily in being niccoliferous instead of manganiferous austenite, and in not containing dissolved cementite. Indeed, these lines and the twinned areas in Fig. 1 might easily be taken for slip bands and twinned areas in copper.

§15. (3) The conditions under which the lines form in manganese steel are the same as those under which the slip bands habitually form in these other metals, and differ radically from those under which twinning usually occurs. These lines like the slip bands result directly from deformation, and thus are mechanical, whereas twinning is rarely mechanical but usually can be induced only by annealing after deformation, except in the extremely soft self-annealing mobile metals such as tin and lead. And though the alleged twins of alpha iron, the Neumann lines, can be caused mechanically and without subsequent annealing, their creation usually requires shock, whereas quiescent stress creates the manganese steel lines as it does the slip bands of all other metals.

§16. (4) So far as we know, slip bands can be developed readily without twinning, but the stress which causes twinning must also cause slip bands, which represent the normal and easy mode of translation. Thus the quiescent deformation which creates the slip bands so abundantly in alpha iron gives no suggestion of Neumann lines, whereas the shock which develops the Neumann lines always develops slip bands also and nearly always abundantly. Even in coarsely crystalline steel in which the Neumann lines form very readily, they are always accompanied by some slip bands. Hence when, as in manganese steel, only a single set of lines forms, analogy indicates that these lines are more likely to be slip bands than twins. This inference is not materially weakened by the fact that certain areas containing Neumann lines may be nearly or quite free from slip bands, and *vice versa.*

§17. (5) The amorphization of the metal along the internal planes of which these manganese steel lines are the outcrops, testified to by the hardening, tallies with the belief that slip has occurred along them but not with the belief that only twinning has. For slip is naturally believed to cause such amorphization, whereas twinning is not, because it implies only rotation into a new orientation without crushing any crystalline matter.

§18. Another consideration, which at first seems to argue that these lines must be slip bands, on examination proves fallacious. That consideration is that in other metals we refer the deformation primarily to slip bands, first because the twinning does not occur during the deformation, and second because for other reasons it seems incompetent to explain the deformation. This might suggest at first that, if the manganese steel lines are all of one kind, they must be slip bands in order to explain the deformation. But these manganese steel lines are competent to explain the deformation even on the theory that they are twins, first because they

form during and not after the deformation, and second because they are numerous enough to account for great deformation. For, if we divide them mentally into the odd and the even numbered, and assume that the odd, for instance, are thicker than the even, then the difference would be cumulative from pair to pair.

§19. The Persistency of these Lines Does not Indicate that they are Twins.—Because in other metals the slip bands have but slight persistency, as already pointed out, whereas twins are much more persistent, we might at first infer from the great persistency of the manganese steel lines that they too are twins. But the radical difference in conditions removes all reason for such an inference. The transitoriness of the slip bands in other metals means that the change which occurs along their slip planes either is not detected by any reagent yet tried, or as Rosenhain holds is effaced quickly on reheating and slowly even in the cold because the amorphous state is here the unstable and the crystalline state the stable one.

But in manganese steel the change from gamma to alpha along the slip planes, proved by the magnetization which accompanies deformation, cannot reverse itself in the cold, because alpha is here the stable form.

The reappearance of these lines or modification of them after gentle reheating we have already referred to the precipitation of cementite along them, and after high heating to the persistence of undiffusing impurities assembled along with the cementite.

§20. Osmond and Cartaud on these Lines.—Against these reasons for regarding these lines as slip bands we have the belief of Osmond and Cartaud[12] that they are twins, apparently because they are persistent, though we cannot be sure that this was the determining cause of that belief. There is the further suggestion that they found these lines related rather to the twins than to the slip bands which form in chrome-nickel steel. We believe that their assertion may be set aside provisionally, not so much because they fail to indicate in what respect these lines resemble twins rather than slip bands, as because they were very probably unaware of the evidence and the reasons outlined in this present paper and especially of the reasons why slip bands in manganese steel should persist.

§21. *In brief*, we believe that these lines may be accepted provisionally as originating in slip rather than in twinning, and as being initially slip bands representing slip along cleavages about which cementite and sonims concentrate on heating.

§22. Summary.—After outlining in Section 1 certain features of plastic deformation bearing on the question at issue, we record in Section 2 that we have developed in this alloy typical twins of the usual kind.

[12] *Journal of the Iron and Steel Institute*, vol. lxxxi (1906, No. III), p. 468.

We outline and in part explain the life history of the lines in manganese steel (Sections 3 and 4). They are apparently persistent (Section 1). The persistency on heating is referable to the concentration of cementite and perhaps of sonims in the very cleavages along which the slip has occurred (Sections 6 and 8). This concentration is of cementite, not of martensite (Section 7).

Both the intersections of these lines at the grain boundaries, and the alternations in the prominence of conjugate sets of these lines, may simulate twinning (Sections 11 and 12).

Indications that these lines are not twins but slip bands are that they begin and develop like slip bands in other metals (Section 13); that they bear to the twins in manganese steel the same relation as regards order of magnitude that the slip bands bear to the twins in other metals (Section 14); that the conditions under which they form are those under which slip bands form in other metals, but unlike those under which twins form (Section 15); that in other metals deformation always causes slip bands but may or may not cause twins, and hence that the fact that these lines always form when manganese steel is plastically deformed whereas the typical twins only rarely do argues by analogy that they are slip bands (Section 16); and that the amorphization which accompanies their formation would naturally accompany the formation of slip bands but not that of twins (Section 17).

It is admitted that twins would be competent to explain the deformation (Section 18).

The fact that twins are far more persistent than slip bands in other metals does not indicate that because these lines are persistent they are twins, because in this alloy their creation leads to the more stable alpha form which cannot revert to gamma, whereas the amorphous metal which forms along the slip planes in other metals and alloys is the less stable, always tending to revert to the crystalline state and thus to efface the slip bands (Section 19).

The opinion of Osmond and Cartaud that these lines are twins is set aside, less because they give no readily controlled reasons for their opinion than because they were probably unaware of the evidence and reasoning here set forth (Section 20).

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Description of Micrographs

Fig. No.	Plate No.	Magnification	Material	Treatment	Description
1	1	500	Mn steel	Metal close to the tensile fracture. Heated to 1.000° for 1 hr, quenched. Polished, hammered, not etched.	Multiple (polysynthetic) annealing twins in greatly deformed manganese steel with typical zigzagging of lines.
2	1	500	Mn steel		Parallel-sided annealing twinned area (4) in greatly deformed manganese steel.
3	1	500	Mn steel	Very small casting (link). Deeply etched.	Manganese steel lines stop short at grain boundaries.
4	1	500	Mn steel		Widmanstättian figuring of manganese steel lines in a casting only 3/64 in. thick in part.
5	1	50	Mn steel	Heated to 1,100°, quenched, strained. Reheated to 575° for 2 hr., etched lightly.	Cementite as Widmanstättian or cleavage figuring and as network in grain boundaries.
6	2	500	Mn steel	Heated to 1,100°, quenched. Polished, lightly strained, not etched.	Alternations in the relative prominence of two conjugate sets of manganese steel lines suggest twinning.
7	2	500	Mn steel	Same as Figs. 1 and 2.	Manganese steel lines stop short or change prominence at grain boundaries.
8	2	50	Mn steel	Heated to 1,100°, quenched. Polished, hammered, not etched.	Manganese steel lines cross the dendrites and probably some grain boundaries without bending.
9	2	500	Mn steel	Same as Fig. 8, reheated to 575° for 2 hr., etched lightly.	Rupture follows the intergranular pro-eutectoid cementite boundaries.
10	3	50	Mn steel	Casting. Polished, hammered, not etched.	The columnar grains in manganese steel made evident on a polished surface by hammering.
11	3	50	Mn steel	Same as Figs. 3 and 4.	The manganese steel lines deflect at grain boundaries so as to suggest twinning.
12	3	90	Mn steel	Extreme end of tensile fracture (copper plated), longitudinal section, etched deeply.	White alpha iron (?) network caused by great deformation, and surrounding dark kernels of gamma iron.
13	3	180	Mn steel	Heated to 1,100°, quenched. Reheated to 575° for 2 hr., quenched. Etched lightly. Hardness = 364.	The manganese steel lines thickened by cementite are nearly or quite parallel in adjoining grains and sometimes cross the grain boundaries.
14	3	500	Mn steel	Same as Fig. 9 before breaking.	Thick coalescence of cementite along the cleavage planes which the manganese steel lines followed.
15	3	500	Mn steel	Same as Fig. 12.	Detail of Fig. 12.

PROGRESS OF DEFORMATION SHOWN ON THE POLISHED SURFACE OF A TRUNCATED WEDGE OF MANGANESE STEEL

Fig. No.	Plate No.	Magnification	Material	Treatment	Description
16	4	180	Mn steel	Heated to 1,100°, quenched, cut into wedge-shaped piece. Polished, squeezed in vise, not etched.	Greatest deformation, lines partly obscured by ruffling.
17	4	500	Mn steel		Less deformation, larger magnification. Many and very prominent lines.
18	4	500	Mn steel		Still less deformation. Two conjugate sets of lines, somewhat curved by faulting (?).
19	4	180	Mn steel		Incipient deformation. A few parallel lines.

PROGRESS OF DEFORMATION SHOWN ON THE POLISHED SURFACE OF A TRUNCATED WEDGE OF COPPER

Fig. No.	Plate No.	Magnification	Material	Treatment	Description
20	4	180	Copper	Rolled bar of pure Lake Superior copper, cut into wedge-shaped piece. Polished, squeezed in vise, not etched.	Great deformation. The lines obscured by ruffling.
21	4	500	Copper		Less deformation, larger magnification. Many typical slip bands, with twin area *abcd*.
22	4	500	Copper		Still less deformation. Two conjugate sets of slip bands.
23	4	180	Copper		Incipient deformation. A few parallel slip bands, and fragment of a grain boundary.

The specimens shown in Figs. 5, 6, 8, 9, 13, 14, and 16 to 19 were cut from cast bars containing 12.8 per cent. of manganese, and 1.24 per cent. of carbon. The specimens shown in Figs. 1, 2, 12 and 15, cut from the tensile test piece, are thought to have the same composition.

Addendum, April 14, 1915. An alternate explanation is that we have to do here with three sets of phenomena, slip bands, and twins of two orders of magnitude; that the deformation lines on the polished surface are slip bands, and the persistent lines found on repolishing and reetching are twins formed along the slip planes. In addition to these twins, which on a repolished surface reproduce the general position which the slip bands have before repolishing, there are the broad twinned bands which give the staircase effect in Figs. 1 and 2.

In Section 5, p. 885, we gave as one reason for the industrial value of this alloy the ease with which it amorphizes. Since then we have found an illustration of this, by comparing the Brinell hardness as found on applying the load all at once in the usual way, with that which results from applying it extremely slowly,[13] and we have applied this test to manganese steel, to the so-called "American ingot iron" with 0.01 per cent. of carbon, or nearly pure ferrite, and to eutectoid steel of 0.92 per cent. of carbon, or nearly pure pearlite.

The deformation starts a hardening process, which, as might be inferred from Muir's classical work,[14] completes itself rather slowly. Hence retarding the Brinell test gives the step in this hardening process, started by each fraction of the deformation caused by the several successive increments of load, time to continue and thus to decrease the indentation which the later successive fractions of the load will cause, and thus in turn to increase the hardness number as found after the whole load has been applied.

The results, condensed here, show that manganese steel increases in hardness nearly 20 per cent. on thus retarding the testing, whereas pearlite, the component of any hypo-eutectoid steel with which manganese steel would have to compete for its special services, does not increase at all. On the other hand the "ingot iron" increased in hardness by no less than 37 per cent. Even after this increase it remains of course extremely soft.

This failure of pearlite to gain hardness on retarding the Brinell test tallies with the observation that undivorced pearlite does not harden greatly in the tensile test as ferrite does. We found that the hardness of a broken tensile test piece, close to the fracture, was about twice as hard as the undeformed part in the case of "ingot iron," and 47 per cent.

[13] In the retarded test a load of 1,600 kg. was applied at 10 a.m. and increased by 200 kg. every 2 hr. till 6 p.m., when it had reached 2,400 kg. At 11 the next morning and every hour thereafter the load was increased by 100 kg., till 5 p.m., when it reached the usual 3,000 kg. It was then left on for the usual time, 30 to 60 seconds. This test was uninterrupted, in the sense that each fraction of the load remained on the specimen continuously from the time when it was first applied till the end of the test.

[14] The Recovery of Iron from Overstrain, *Philosophical Transactions of the Royal Society*, vol. cxciii, A, p. 1, and especially p. 13.

harder than the undeformed part in the case of steel of 0.11 per cent. of carbon,[15] but the increase was very slight in the case of steel of 0.92 per cent. of carbon, when its pearlite was undivorced.

Increase of the Brinell Hardness Caused by Retarding the Brinell Test

Material	Brinell Hardness		Percentage Increase Caused by the Retardation
	Under the Usual Direct Test	Under a Test Greatly Retarded	
Hadfield's manganese steel quenched from 1,100°.	170	202	19
"American ingot iron," carbon 0.01 per cent.	73	99	37
Eutectoid steel, carbon 0.92 per cent., air cooled from 900°.	259	259	0

Discussion

J. E. Stead, Middlesbrough, England (communication to the Secretary*).—As I have not personally studied the question of quenched and distorted manganese steels I cannot give any information of my own on the subject.

The question as to whether certain indications in strained manganese steel are "slip bands" or "twins" might be settled, I think, by heat tinting—a method I have applied with success to certain meteorites and other alpha irons in which there were undoubted twins. After polishing and very slightly etching with picric acid in alcohol, and then "heat tinting," the colors assumed by the twinned strata and the adjoining metal are different. The reason of this, as I have proved repeatedly, is that different faces of the iron crystals are differently attacked by oxidizing and other reagents, and consequently some of the faces take temper oxidation tints more rapidly than others. This simple method enables one to determine with certainty whether the "twins" are actually twins and differently oriented from the bed in which they are found. After correspondence with Doctor Howe, I think we both agree that Neumann lines should not be called "lines" but "bands" or "strata," preferably the latter term, as I have found that they are more or less continuous throughout the crystals, and it would be just as improper to call the edges of cementite plate by the term "lines" as to call stratified twins by the same term.

The observation that on annealing strained austenite manganese steel, cementite settles out along the slip planes, is most interesting and is a quite original observation.

[15] *Proceedings of the American Society for Testing Materials*, vol. xiv, pt. 2, p. 28 (1914).

* Received Mar. 26, 1915.

Manganese Steel

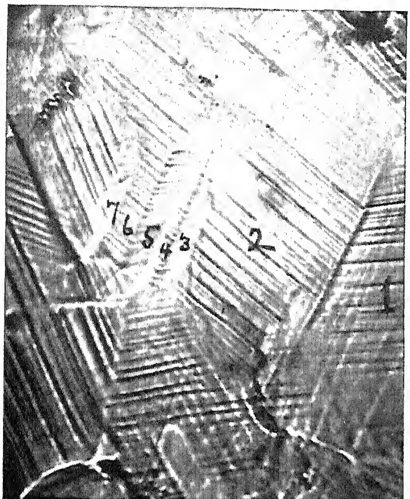
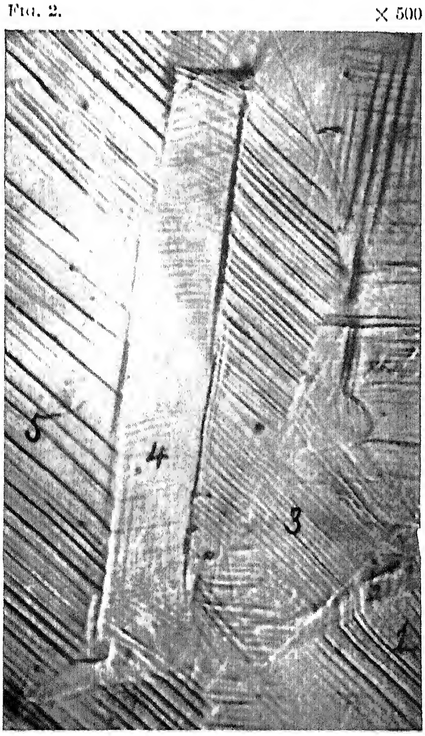


FIG. 1. × 500

Twins in Mn steel, greatly deformed, annealed, polished, and hammered. Not etched.

FIG. 2. × 500



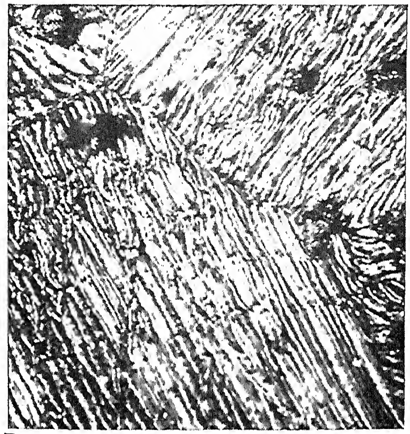
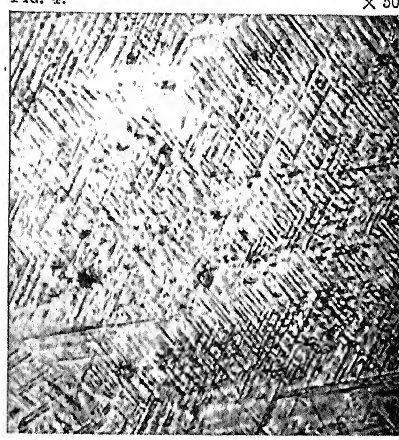


FIG. 3. × 500

Mn steel lines in one and the same casting, very deeply etched.

FIG. 4. × 500



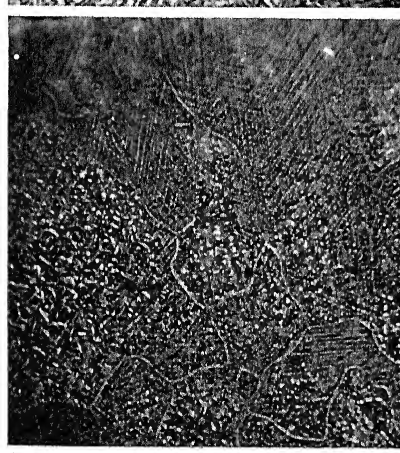


FIG. 5.—Cementite in Mn steel, strained, heated to 575° C., polished, and lightly etched. × 50.

Manganese Steel

Fig. 6.—Lines in lightly strained Mn steel, simulating twins. Not etched. × 500.

Fig. 8.—Lines in Mn steel severely strained, crossing the dendrites. Not etched. × 50.

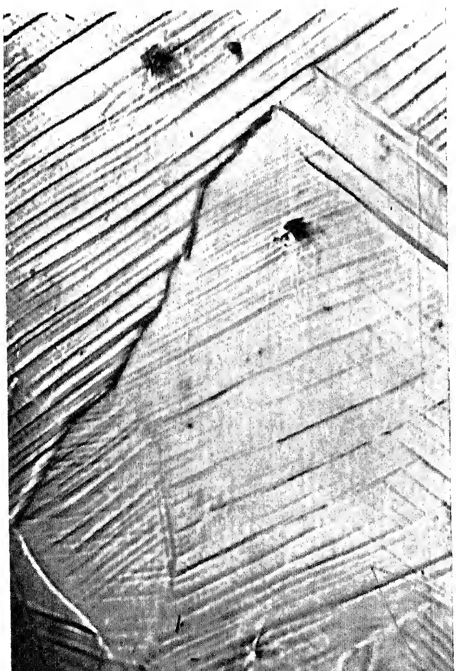


Fig. 7.—Lines in Mn steel stop at grain boundaries. Not etched. × 500.

Fig. 9.—Rupture follows the cementitein Mn steel reheated to 575° C. Etched lightly. × 500.

Manganese Steel

FIG. 10.—Columnar crystals disclosed by hammering an unetched polished surface. × 50.

FIG. 13.—Mn steel lines cross grain boundaries. Same treatment as Fig. 14. Etched lightly. × 180.

FIG. 11.—Small casting strained and etched deeply. × 50.

FIG. 14.—Cementite along slip planes. Strained. heated to 575° C., etched lighlty. × 500.

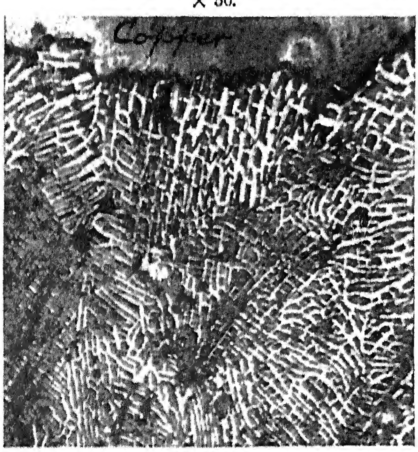


FIG. 12. × 90.

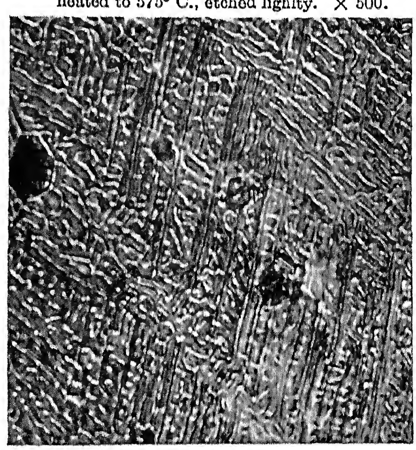


FIG. 15. × 500

Longitudinal section through copper-plated tensile fracture. Etched deeply.

PLATE 4

Progress of the development of the lines in Mn steel and of the slip bands in copper.

Manganese Steel | Copper

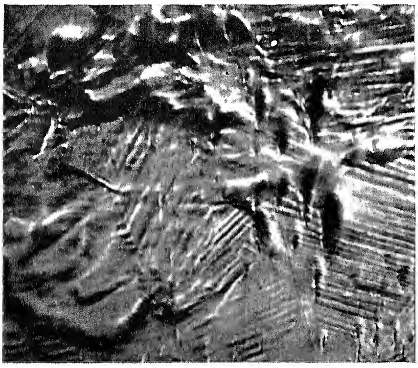


Fig. 16.

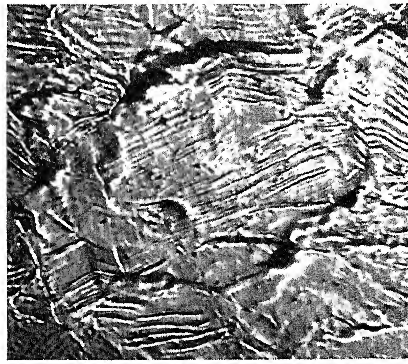


Fig. 20.

Great deformation. × 180.

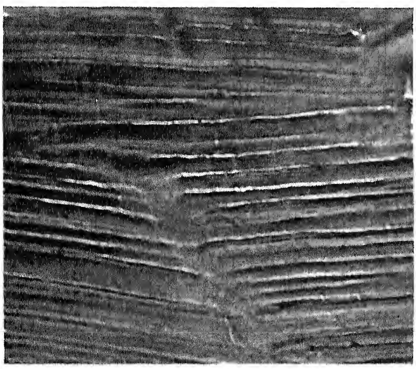


Fig. 17.

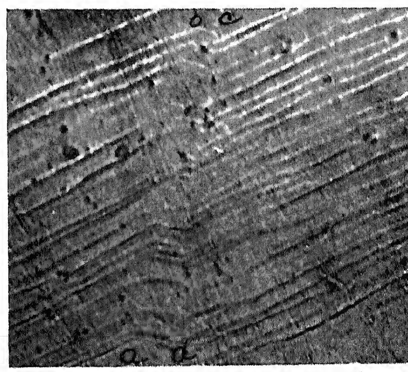


Fig. 21.

Less deformation. × 500.

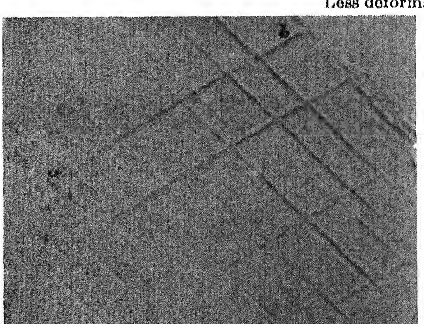


Fig. 18.

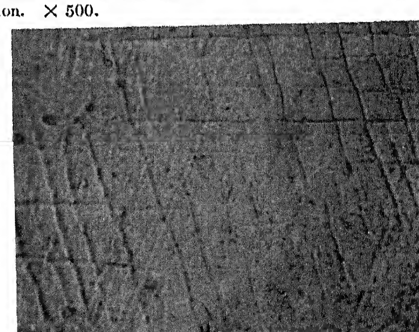


Fig. 22.

Still less deformation. × 500.

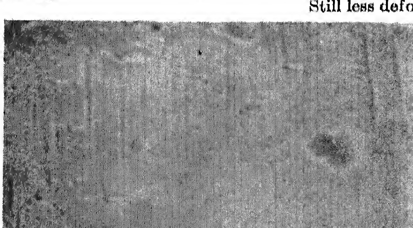


Fig. 19.

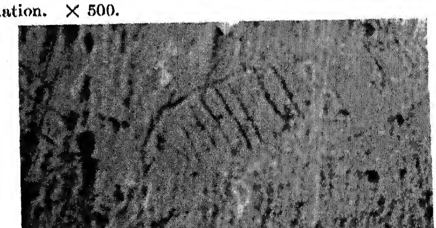


Fig. 23.

Least deformation. × 180.

Structure and Hysteresis Loss in Medium-Carbon Steel

BY F. C. LANGENBERG,* CAMBRIDGE, MASS., AND R. G. WEBBER,† ATHENS, OHIO

(New York Meeting, February, 1915)

During the course of some magnetic investigations which the authors have under way, six bars of 0.43-carbon steel were tested, a permeameter designed after the Hopkinson yoke type being used. The results obtained were of some interest and although far from complete serve at least to show the necessity for a careful investigation of the previous history of the samples under study as well as the chemical composition.

A steel of given chemical composition may assume a great variety of structures, depending on its treatment. Professor Sauveur has devised a very clear and effective means of illustrating this point. The accompanying microphotographs show some of the possible structures assumed by a 0.30 carbon steel (Fig. 1).

It is now a well-established fact that a structural change is accompanied by changes in the tensile strength, elastic limit, hardness, etc., and it seems reasonable to assume also, a change in the magnetic properties.

Six bars of a 0.43-carbon steel were used, all the bars being taken from the same rod and treated as follows:

Bar 1. Heated to 1,100° and cooled in the furnace.
Bar 2. Heated to 1,000° and cooled in the furnace.
Bar 3. Heated to 900° and cooled in the furnace.
Bar 4. Heated to 1,000° and cooled in air.
Bar 5. Heated to 900° and cooled in air.
Bar 6. Untreated.

The bars were then turned to 13 mm. diameter and tested by the Hopkinson yoke method. After testing, three specimens were taken from each bar, one from each end and one from the middle. The homogeneity of the bars was tested in this manner and in no instance could any difference in structure in the three specimens[1] be detected.

Bars 1 to 3 inclusive are pearlitic in structure. The white areas are ferrite (iron with certain impurities in solution) and the dark areas pearl-

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† Instructor in Physics and Electrical Engineering, Ohio University.

[1] Specimens were prepared by Charles Frohnert.

ite (the eutectoid of Fe_3C and ferrite). Bar 1 has a coarser structure than No. 2, and No. 2 in turn a coarser structure than No. 3. No. 6 is also pearlitic and is similar in every way to Nos. 1, 2, 3, except that its structure is very fine. No. 6 was the untreated specimen and represents the structure of the material as it came from the rolls. Nos. 4 and 5 have an entirely different structure. On passing through the critical range on cooling the ferrite has been rejected to the boundaries of the grains and the

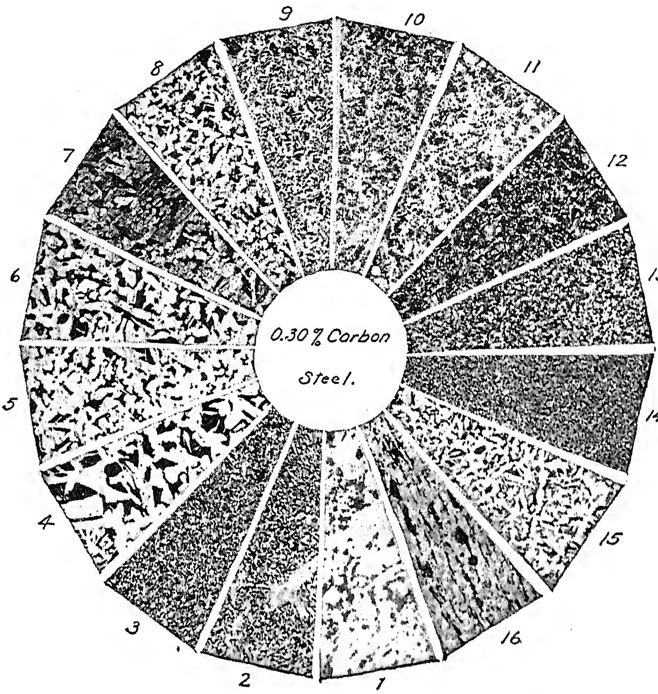


Fig. 1.—Various Structures of Mild Steel (0.30 Per Cent. C). (A. Sauveur.)

1. Steel as cast.
2. Steel cast and imperfectly annealed (remnants of ingotism).
3. Steel cast and properly annealed.
4–8. Heated to various temperatures above critical range for various lengths of time and slowly cooled in furnace. Ferrito-pearlitic structures of different degrees of coarseness.
9–13. Heated above critical range, followed by cooling in air or oil, or heated above critical range, cooled in water or oil and reheated to 600° C. Ferrito-sorbitic or ferrito-sorbito-troostitic structures.
14. Forged and finished at low temperatures.
15. Forged and finished at high temperatures.
16. Cold worked.

dark portions in these samples are a mixture of pearlite and sorbite. Sorbite can be regarded as poorly defined pearlite and is one of the transition constituents between austenite, existing above the critical range, and pearlite, below the range. No. 4 is more sorbitic than No. 5 and also has a larger grain.

Below each microphotograph (Figs. 2 to 7) is placed the B–H curve for the specimen together with some of the data which we regard as significant.

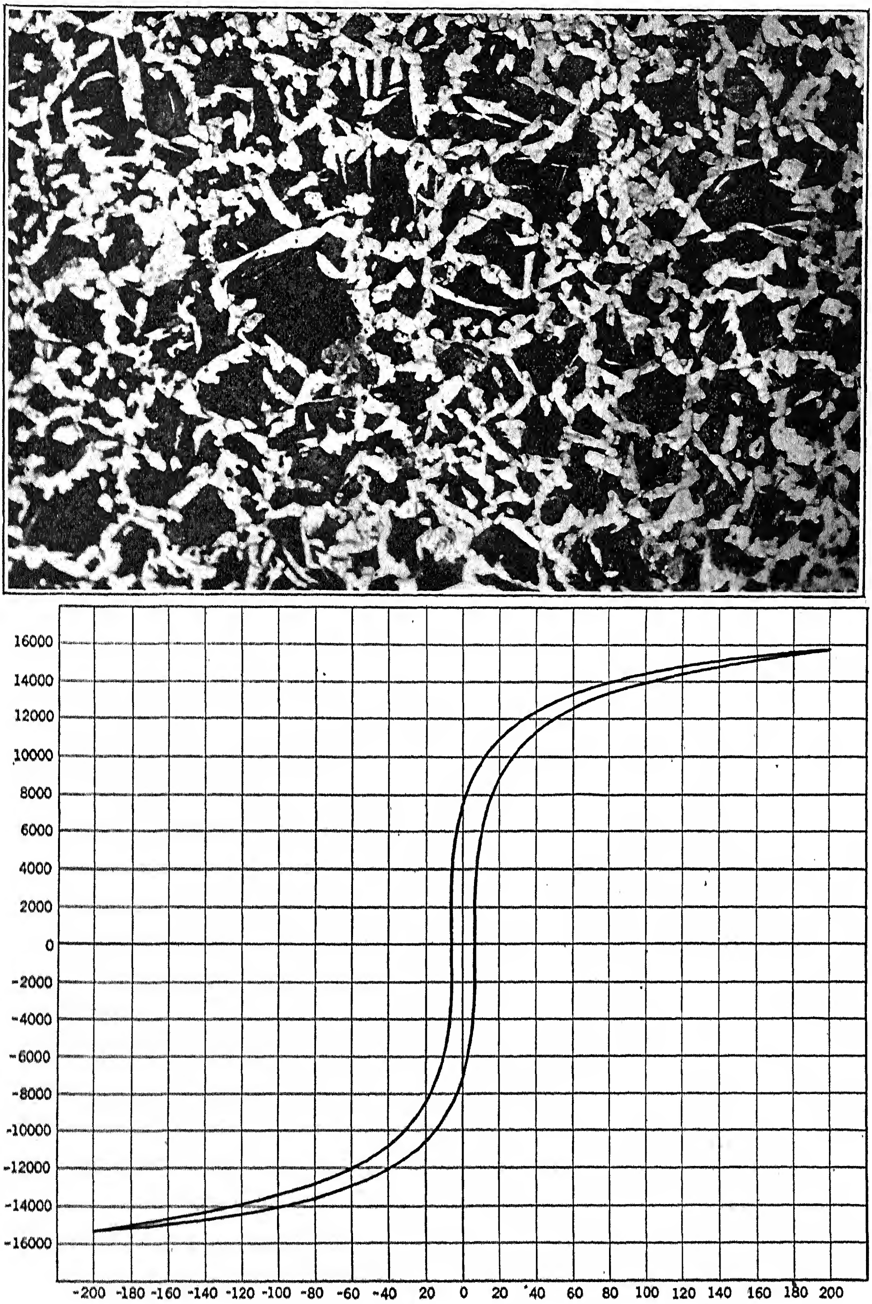


FIG. 2.—BAR 1.

Hysteresis Loss, 17,280
Residual B, 6,700
Coercive Force, 3.65
Hardness,

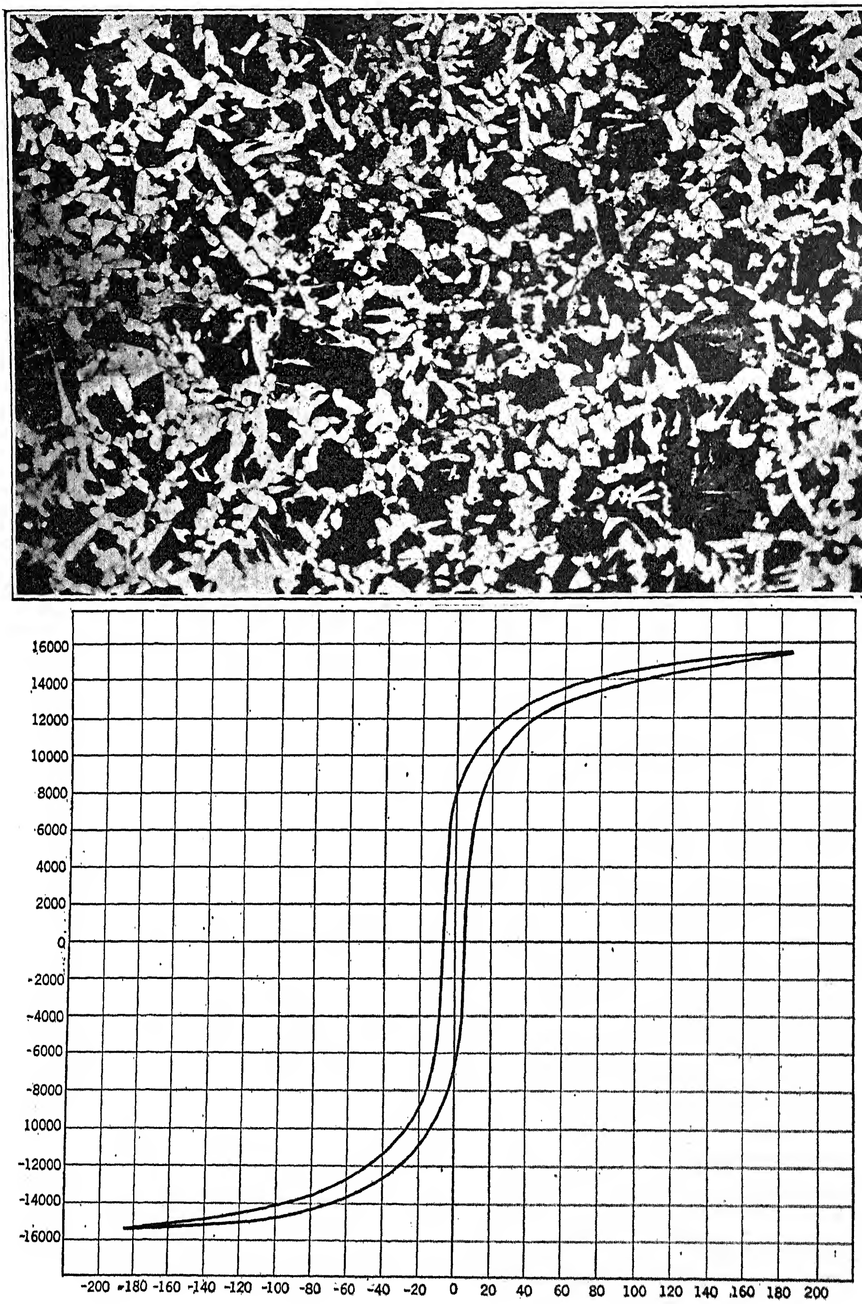
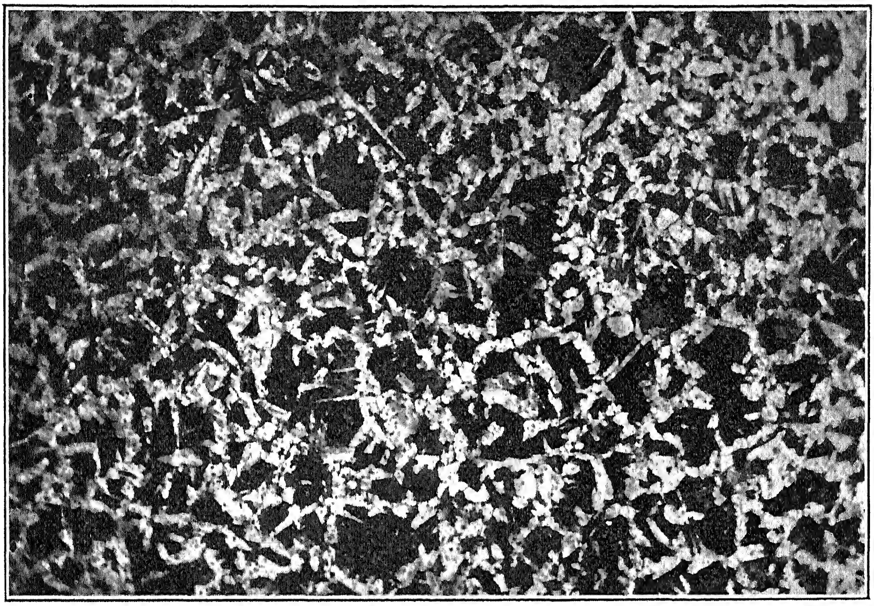


FIG. 3.—BAR 2.

Hysteresis Loss,	18,240
Residual B,	6,800
Coercive Force,	3.70
Hardness,	131



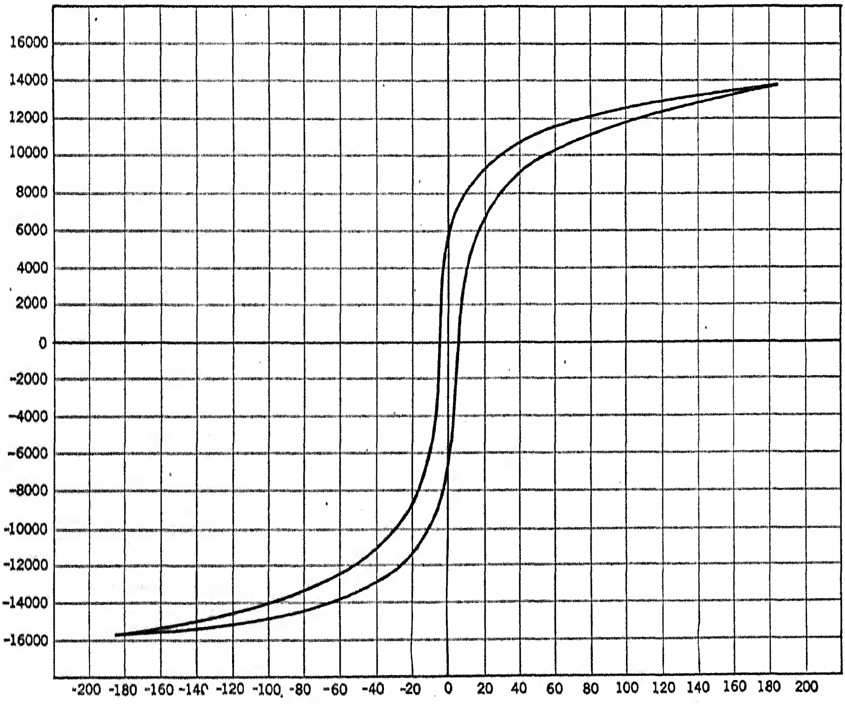


FIG. 4.—BAR 3.

Hysteresis Loss. 21,920
Residual B, 7,000
Coercive Force, 3.72
Hardness, 131

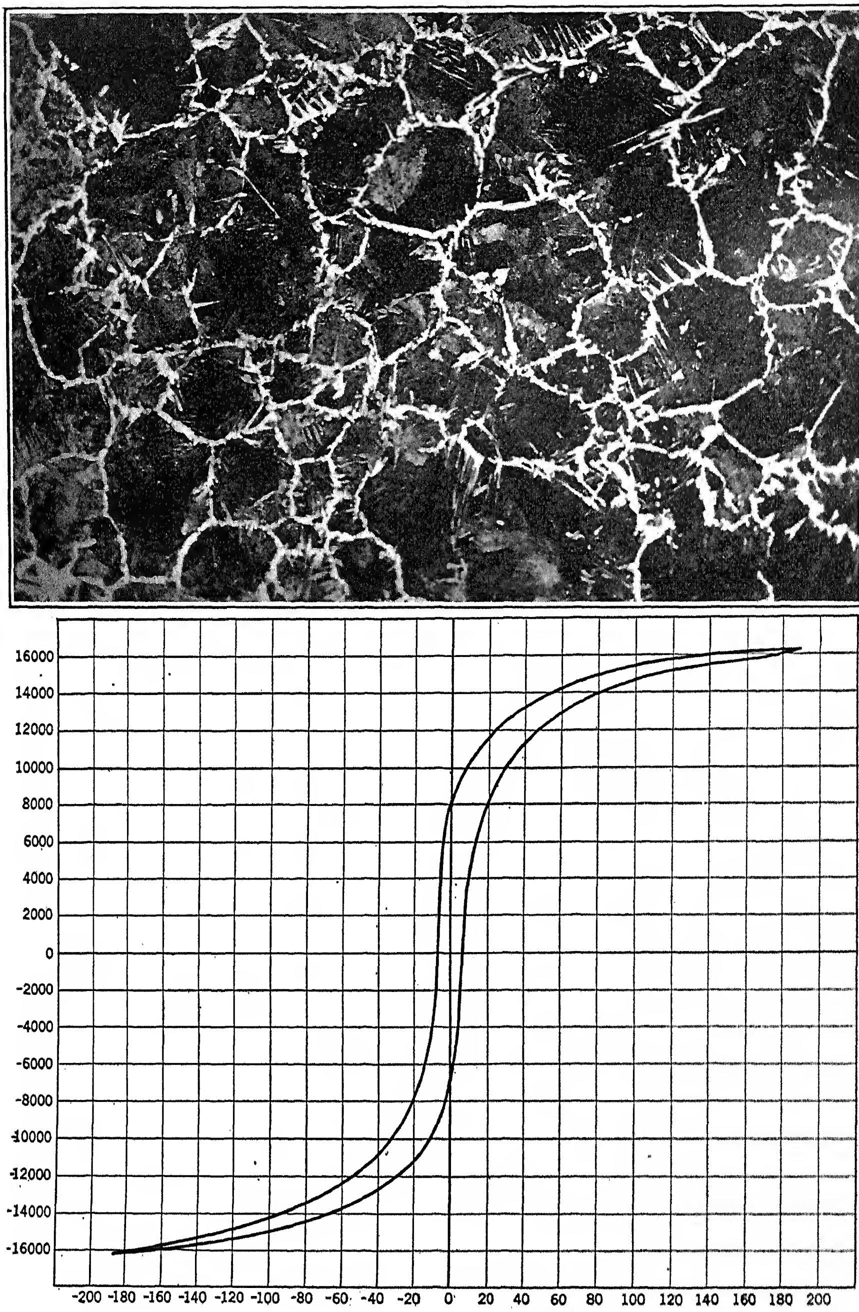


Fig. 5.—Bar 4.

Hysteresis Loss,	26,240
Residual B,	7,700
Coercive Force,	7.00
Hardness,	140

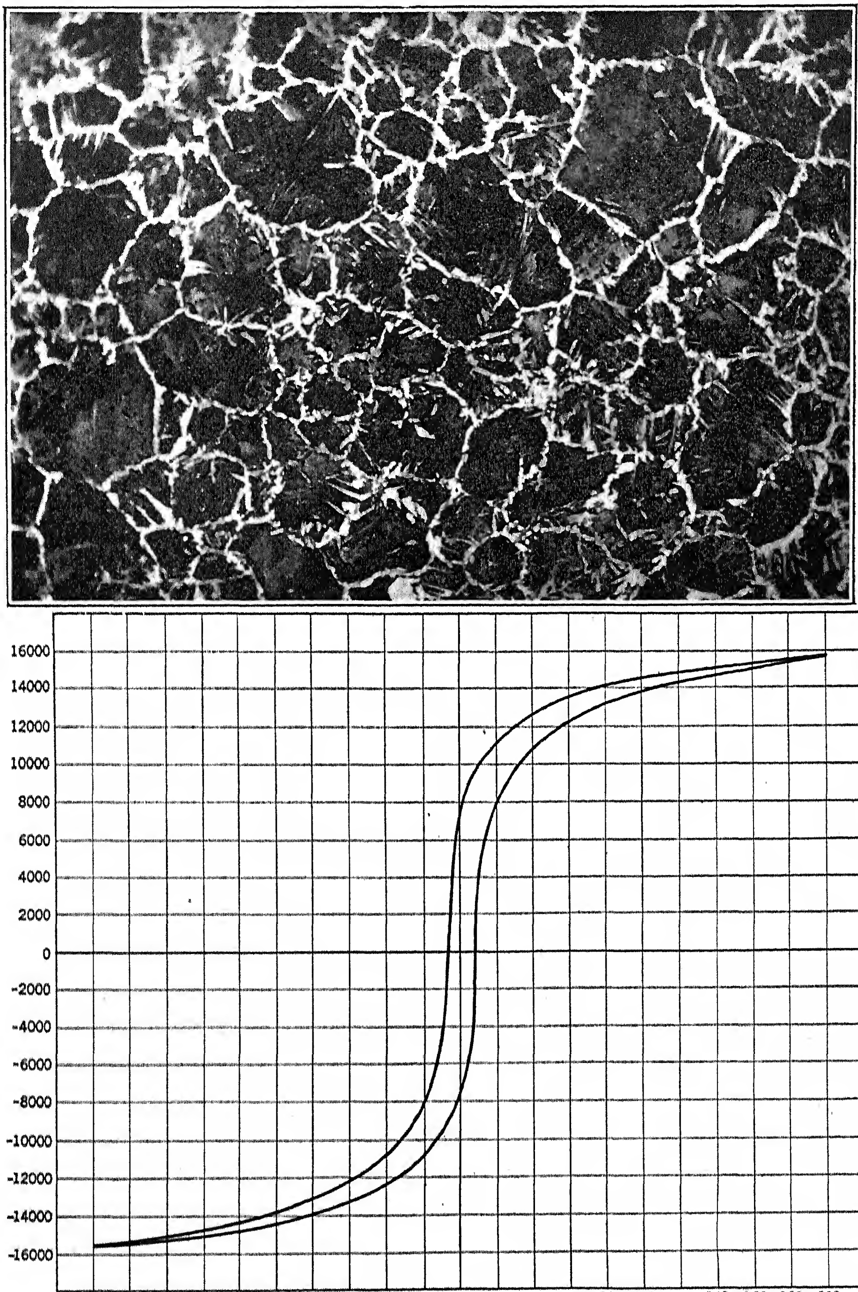


FIG. 6.—BAR 5.

Hysteresis Loss, 25,760
Residual B, 7,300
Coercive Force, 7.00
Hardness, 131

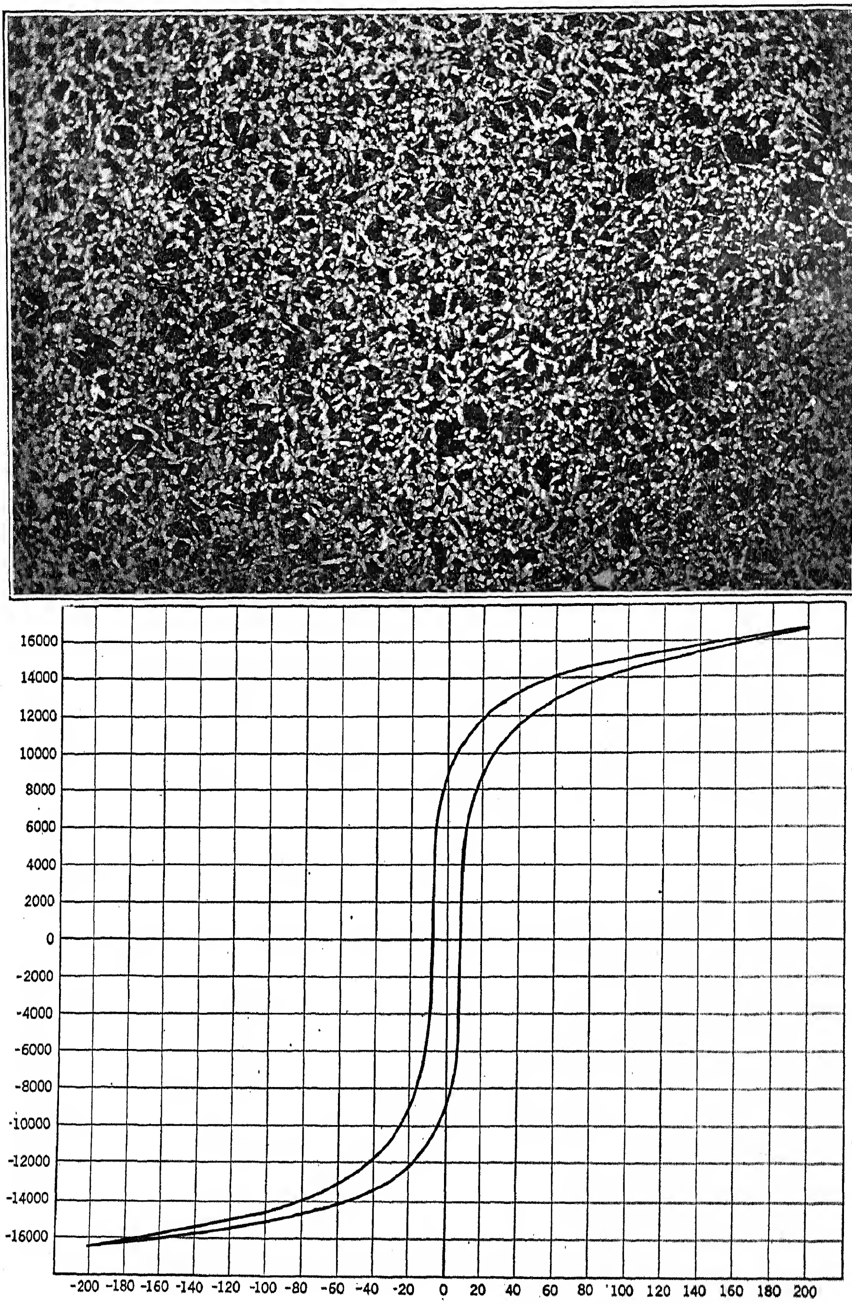


FIG. 7.—BAR 6.

Hysteresis Loss,	29,120
Residual B,	9,700
Coercive Force,	7.50
Hardness,	131

Data

Bar No.	Loss in ergs per c.c.	Residual B	Coercive force	Brinell hardness
1	17,280	6,700	3.65	131
2	18,240	6,800	3.70	131
3	21,920	7,000	3.72	131
4	26,240	7,700	7.00	140
5	25,760	7,300	7.00	131
6	29,120	9,700	7.50	131

The hysteresis loss was determined as follows: The area of the loop inclosed by the B–H curve was measured by a planimeter. The coordinates being in absolute units, the resulting area as determined by the planimeter was multiplied by our scale constant, which was 40,000, and divided by 4π.

Residual curves were not determined, but for each bar the residual induction B was determined after the magnetizing force H had reached its maximum. The coercive force was determined by measuring the magnetizing force H necessary to reduce residual B to zero.

Hardness of the specimen was measured by a Brinell machine and results are expressed in Brinell hardness factors.

Bars 1, 2, and 3 present a very interesting study. The only variable factors in these three bars are the relative size of grain and the coarseness of the striations of pearlite. Bar 1 has a large grain and the pearlite is coarsely laminated. Its hysteresis loss per cubic centimeter was 17,280. Bar 2 has a finer structure than No. 1, and shows a hysteresis loss of 18,240. Bar 3 is still finer than bar 2 and shows a loss of 21,920.

Bar 6, as before stated, is also pearlitic and differs from Nos. 1, 2, and 3 in having a much finer structure. The hysteresis loss in this bar rises to 29,120 ergs per cubic centimeter, which is 68 per cent. greater than the loss in bar 1. It is to be borne in mind that both are in the unhardened condition.

Bars 4 and 5 show an entirely different structure to any of the other bars. Bar 4 has a larger grain than No. 5 but the treatment employed to give the bar its coarse structure also produced a larger proportion of sorbite than is present in Bar 5. Bar 5 has a hysteresis loss 480 ergs less than bar 4, which seems contradictory to the results shown by the previous bars, as bar 5 has a finer structure than bar 4. The finer structure of bar 5, however, is only apparent and not real. The gross structure is finer—that is, the grains are finer—but the structure as revealed under higher magnification shows that bar 4 is composed of sorbite, laminated pearlite being almost entirely absent. In bar 5 considerable laminated pearlite is present.

Summary

1. Bars tested, six in number, carbon content 0.43 per cent. Nos. 1, 2, 3, and 6 were pearlitic and Nos. 4 and 5 were sorbitic.

2. With decreasing grain size Nos. 1, 2, 3, and 6 show a rise in their hysteresis loss, increasing residual B, and increasing coercive force.

3. Bar 4, comparable in grain size to bar 1, but differing from bar 1 in being sorbitic whereas No. 1 is pearlitic, shows an increase in its hysteresis loss of approximately 50 per cent.

4. In magnetic testing careful attention must be given to the internal structure of the metal undergoing test. A mere statement of its condition as hardened or unhardened, annealed or tempered, is idle and often misleading.

5. Our results lead us to infer that a minimum hysteresis loss could be obtained by the combination of large grains with coarsely laminated pearlite.

Discussion

Henry M. Howe, Bedford Hills, N. Y.—The results here given are of great interest and are welcome as forming another rivet to increase the stability of our present theories of the constitution and properties of steel.

The important discovery is that the hysteresis increases with the fineness of the structure, that is to say with the extent of contact between the particles of ferrite and cementite. All of the specimens except No. 4 consist of ferrite and cementite of different degrees of intimacy of admixture.

If hysteresis represents something akin to surface friction, which retards the rotation of the ferrite on the application and release of the magnetizing force, then it is only natural that it should increase as it does with the intimacy of admixture, because it is at the surface of contact of a given particle of ferrite with the neighboring particles of cementite that this friction is to be expected, and of course the extent of surface must increase with the intimacy of admixture.

The somewhat greater hardness of specimen 4 than of the others may perhaps be referred to the principle discussed in my paper, Why Does Lag Increase with the Temperature from Which Cooling Starts?[2] The furnace-cooled specimens were cooled so slowly that the influence of the initial temperature of heating is effaced. But in the air-cooled specimens we have the suggestion that the high temperature, 1,000°, from which the cooling of No. 4 started has prevented the completion of the transformation, with the result that the metal is slightly harder than any of the others.

The difference is not to be explained by difference in the fineness of structure, because the whole range of fineness from No. 1 to No. 6 is accompanied by no change in hardness, the only change being an appreciable increase of hardness following the air cooling from 1,000°.

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[2] *Trans.*, xlv, p. 516 (1913).

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[NOTE.—In this Index the names of authors of papers are printed in small capitals, and the titles of papers in italics. Casual notices, giving but little information, are usually indicated by bracketed page-numbers. The titles of papers presented, but not printed in this volume, are followed by bracketed page-numbers only.]